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American Institute of Mining and Metallurgical Engineers

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YEAR BOOK

This Volume Contains Papers and Discussions Presented at Meetings Held in San Francisco, October, 1929, and New York, February, 1930; Officers, Committees, Abstracts, Titles of Papers in 1930 Volumes, List of 1930 Technical Publications and Consolidated Index

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PREFACE

This, which constitutes the Year Book for 1930, closes the series of TRANSACTIONS for the year. Five special volumes have been previously issued, as below:

TRANSACTIONS, PETROLEUM DEVELOPMENT AND TECHNOLOGY, 1930.
TRANSACTIONS, COAL DIVISION, 1930.
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In this volume will be found the records of the year with lists of officers, references to meetings, necrology and similar material. There is also included a consolidated index of publications and a classified series of abstracts. By using these any member can easily find any article published within the year by the Institute and determine its scope and character. He can then, if more details are desired, turn to the special volume in which it appears on the shelves of his own or any convenient public library.

Finally, there is included in this volume a series of technical papers on metal mining and nonferrous metallurgy, these being the subjects that are of interest to the largest group of Institute members. The contents of the other volumes are given on later pages.

Each member is entitled to one volume of TRANSACTIONS in addition to the Year Book volume; the others will be supplied on request at the cost price, $2.50 per volume. All are now supplied in cloth binding without extra charge.

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California, University of, Berkeley, Cal	Students Mining Association	Philip K. Strong	Alan T. Wilson
Carnegie Inst. of Technology, Pittsburgh, Pa	The Miners and Metallurgists Society	William Walk	Archibald Jones, Jr.
Case School of Applied Science, Cleveland, Ohio	Pick and Shovel Club	R. H. Randolph	W. B. Scott
Colorado School of Mines, Golden, Colo	Student Branch of A. I. M. E.	Donald Peaker	T. H. Evans Jones
Harvard University, Cambridge, Mass	Geological Club	H. E. Quinn	Lincoln Thiesmeyer
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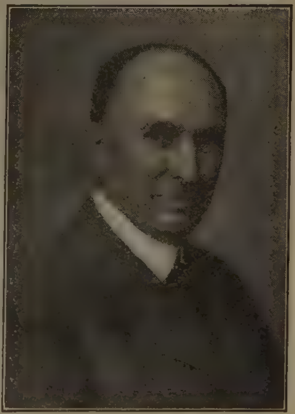
Seeley W. Mudd Memorial Fund

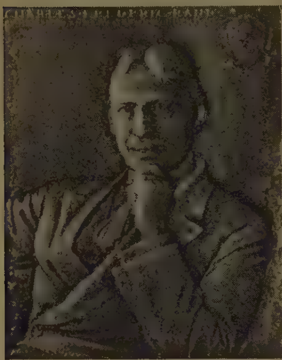


THE Seeley W. Mudd Memorial Fund, consisting of $100,000, was given the Institute in 1929 by the family of Colonel Mudd who had served the Institute as Director, Vice-president and on many important committees. His long career of distinguished professional and public service is fittingly commemorated by this fund. The income is available for support of research and other special purposes. In view of Colonel Mudd's life work with the nonferrous metals and his keen interest in the younger men of the profession it is purposed to apply the fund particularly to the benefit of these two. Control of income is exercised by the Board of Directors on recommendation of a Committee consisting of the President and Secretary, ex officiis, together with three others selected by the Board. The Committee consists at present of the following members: W. H. Bassett, President; H. Foster Bain, Secretary; Harvey S. Mudd serving until October, 1931; Edgar Rickard serving till October, 1932; George Otis Smith serving until October, 1933.

On recommendation of this Committee the Board has authorized as a first project the preparation of a series of small books designed especially to be helpful to the Junior Members. These are now being written and it is hoped will be available for distribution in 1931. The list includes: A Brief History of American Mining, by T. A. Rickard; Mining Costs, Why and What, by Arthur Notman; Introduction to Mineral Economics, by H. Foster Bain and T. T. Read; Choice of Methods—A Record of Experience in Making Decisions, with chapters by W. H. Bassett, F. W. Bradley, E. DeGolyer, Howard N. Eavenson, A. B. Parsons, J. V. W. Reynders, L. D. Ricketts, Robert E. Tally, Pope Yeatman and others. One other book is projected but final selection has not been made. The purpose of these books is to afford the younger men who now come into a highly specialized profession some substitute for the broad

contacts their predecessors had in their formative years. It is believed that in this way Colonel Mudd's ideal of handing on to our successors the best that experience has taught us will be appropriately carried out, but other projects will be considered by the Committee in due course.

Charles F. Rand Foundation


Medallion by Anthony de Francisci

FRIENDS of the late Charles F. Rand presented in 1930 a sum of money from which the income is to to be available to support various phases of the work of the Institute in which Mr. Rand was so deeply interested. The terms of the deed of gift are sufficiently broad to permit of a wide range of activities as shall from time to time best serve to commemorate the life and services of Mr. Rand, Past President, Director and long-time Treasurer of the Institute. The specific terms of the deed of gift are as follows:

"The fund collected to establish a memorial to the late Charles F. Rand shall be placed in the custody of the Board of Directors of the American Institute of Mining and Metallurgical Engineers, to be set up as a trust to be known as the 'Charles F. Rand Foundation,' the income of which shall each year be applied to a distinct purpose to be decided upon in that year by vote of the Board of Directors. This purpose shall be such as to promote the general welfare of the Institute and to constitute a permanent memorial of the usefulness of Mr. Rand, who for many years gave unstintingly of his time and talents in its service. The Board may also at its discretion allow the income to accumulate.

"It shall be within the discretion of the Board of Directors to establish a medal to be known as the Charles F. Rand Memorial Medal to be awarded at such a time and under such rules as may be determined by the Board of Directors, for Distinguished Achievement in Mining Administration."

The members of the Rand Memorial Committee consist of Karl Eilers, Chairman, Walter H. Aldridge, Arthur S. Dwight, J. V. W. Reynders and B. B. Thayer. Steps are being taken to provide for the medal mentioned above.

Rocky Mountain Fund

THE Rocky Mountain Club was established in 1907 to keep alive the spirit of the West among the men of the mountains and the plains whose business necessitated their residence in the East. The annual dinners, when the East met the West again, soon became famous and funds were collected for a club house. True to the motto of the Club these funds were, in the main, given instead during the war for use in various patriotic activities. In 1928, recognizing the community of interest between the Club and the Institute the two were merged, a special class of membership being established in the Institute for Rocky Mountain members. The residue of the funds of the Club, amounting to a little more than $100,000, were used to establish a fund the income from which should be available for research or other activities of especial interest to the Rocky Mountain region. The funds are administered by the Board on the advice of a Committee consisting of Walter H. Aldridge, Chairman, whose term expires Jan. 1, 1931; Harvey S. Mudd, who serves till January, 1932; George Otis Smith, serving until 1933.

Three projects have been approved and financed from the income of this fund.

1. The first Reverberatory Furnace Conference, modeled along the line of the very successful Open Hearth Conferences established by the Institute among steelworkers in 1926, was held at Salt Lake City in May of this year. Representatives of 10 copper smelting companies met under the auspices of the Institute and exchanged experiences around the conference table. A record of the proceedings was furnished to each and to the companies recommended, with the compliments of the Rocky Mountain Fund.

2. A history of the development of the so-called "porphyry" copper mines, with especial emphasis on the social and economic significance of this great achievement, is being prepared by A. B. Parsons with the active cooperation of the officers and engineers of the companies concerned. It is believed that this building up of a great mass production movement from material previously below the limit of availability constitutes one of the most brilliant chapters in the history of mining in the Rocky Mountain region and that publication in one place of this record will do much to increase the prestige of miners and metallurgists. The book will constitute the first of a series of Rocky Mountain Monographs it is hoped to publish.

3. A monographic account of the principal contributions to the knowledge of ore deposits made in the last quarter century, with illustrations from the districts of the Rocky Mountain region, has been authorized. John Wellington Finch is serving as chairman of the special committee charged with preparation of this volume, which will be dedicated to Waldemar Lindgren whose studies personally and through students and associates have done so much to elucidate the problems of the region. It is expected to be the second of the Rocky Mountain Monographs and also to serve as a background for the excursion guides issued in connection with the International Geological Congress in 1932.

Medals and Awards

The Institute is custodian of funds for support of numerous gold medals and prizes and has representatives on boards awarding still others. Details regarding the Institute Awards are given below.

The Institute is represented on the boards of the John Fritz Medal, the Hoover Medal and the Washington Award, in connection with the three national societies of Civil, Mechanical and Electrical engineering; and, also, in the case of the Washington Award, with the Western Society of Engineers.

Robert W. Hunt Medal and Prize

THE partners of Robert W. Hunt established a fund which was presented to the Institute at a fitting ceremony on May 27, 1920, to establish an annual medal and a sum of money to be awarded under the following rules:

1. The award shall consist of two prizes: first of a gold medal, second of a sum of money; a certificate to accompany each prize. The money prize shall not be awarded to a member over 40 years of age, but under unusual circumstances, both prizes may be allotted to one person provided that he is not over 40 years of age. In general it will be understood that the Committee shall award the money prize to the younger men, rather than the medal.

2. The awards shall be made not oftener than once a year to that person or persons contributing to the Institute the best original paper or papers on iron and steel. The scope of the term "iron and steel" shall be determined by the subcommittee considering the awards. In general, papers dealing with the practical side of the subject have preference over those dealing with the theoretical side in recognition of the fact that Captain Hunt's main contributions to the industry have been in the improvement of production and quality of material.

3. A subcommittee of three to five, including the Chairman of the Iron and Steel Division, shall be appointed by the Iron and Steel Division annually to adjudge the award subject to approval.

4. The awarding committee shall submit its report to the Iron and Steel Committee at its October meeting, and the award shall be certified to the Secretary of the Institute in time to permit the presentation to be made at the Annual Meeting of the Institute.

5. The recipient of the award shall be designated "The Hunt Medalist."

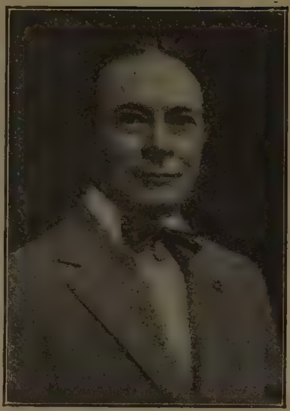
Awards of the medal have been made as follows:

1920—Robert Woolston Hunt.
1926—Charles Lewis Kinney, Jr.
1928—John A. Mathews.
1929—Edgar Collins Bain.
1930—James Aston.

Awards of the prize have been made as follows:

1928—C. H. Herty, Jr.
1929—William E. Griffiths.

J. E. Johnson, Jr. Award



THIS award is made from the income of a fund of $3000 donated by Mrs. Margaret Hilles Johnson in memory of her husband, J. E. Johnson, Jr., who was a prominent engineer, author of two valuable volumes on iron blast-furnace construction and practice, vice-chairman of the Institute's Iron and Steel Committee, and a frequent contributor of papers to the Institute's TRANSACTIONS. It is administered by the Iron and Steel Division.

The intent of the donor is to encourage young men in creative work in branches of the metallurgy or manufacture of pig iron with which the professional activities of Mr. Johnson were chiefly concerned. The control of the fund and distribution of the awards having been vested in the Board of Directors of the Institute, this Board has made the following regulations concerning it:

(1) The award shall not be made to persons over forty years of age.

(2) The award shall not be made oftener than once a year.

(3) The award shall be accompanied by a certificate suitable for framing.

(4) The award shall be made to some promising engineer, selected because of a meritorious research, invention, or contribution to the professional literature of iron and steel, along the lines with which the professional activities of Mr. J. E. Johnson, Jr. were chiefly concerned.

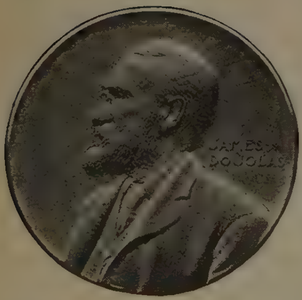
(5) The Iron and Steel Division of the Institute shall recommend to the Board of Directors before the Annual Meeting of the Institute, each year, any suitable person qualified to receive the award.

(6) The Board of Directors of the Institute may change these regulations as may be necessary or advisable within the spirit and intent of the donor's desire to bring encouragement and distinction to promising young engineers.

This prize has been awarded as follows:

1923—Alexander L. Feild.
1926—Selwyne Perez Kinney.
1927—Thomas L. Joseph.
1928—P. H. Royster.
1929—No award.
1930—William S. Unger.

The James Douglas Medal



ENDOWMENT for an annual gold medal, commemorating Dr. James Douglas, twice President of the Institute, was established by anonymous donors in 1922. The rules governing the award of the medal, as adopted by the Board of Directors, include, the following main provisions:

1. The metal to be awarded once a year, preferably at the time of the annual meeting. The medal is to be accompanied in every case by an engrossed diploma containing a citation of the service or achievement upon which the award is based.

2. Eligibiliy. (*a*) Distinguished achievement in nonferrous metallurgy. (*b*) No limitations as to nationality, membership in the Institute or otherwise. (*c*) Recipient must be a living person, able and willing to

present himself in person to receive the award at the time and place to be prescribed by the Board of Directors.

3. The choice of a recipient is to be based upon a report by a committee of fifteen members of the Institute interested in nonferrous metallurgy, chosen by the Board of Directors and distributed as widely as possible geographically. The tenure of office shall be for three years, so arranged that a portion of the Committee shall be elected each year. A member shall not be eligible for reappointment to the Committee for a period of one year after his term has expired. The President shall be, ex officio, a member of the Committee but not its chairman. The award shall finally be made by a majority vote of the Board of Directors, after a full consideration of the report by the Committee. The Directors shall have the right to reject the report of the Committee.

4. The Committee may call on the membership for nominations, but such nominations must be accompanied by a detailed argument in favor of the award and details sufficient to permit a proper citation to be drawn.

Awards of the medal have been made as follows:

1923—Frederick Laist.
1924—Charles Washington Merrill.
1925—William Hastings Bassett.
1926—John Michael Callow.
1927—Zay Jeffries.
1928—Selwyn G. Blaylock.
1929—Paul Dyer Merica.
1930—John Van Nostrand Dorr.

The William Lawrence Saunders Gold Medal

THE William Lawrence Saunders Gold Medal for achievement in mining was established in 1927. The rules adopted for the award of this medal are as follows:

1. The Mining Medal may be awarded once a year, but not oftener. The medal is to be accompanied by an engraved diploma containing a citation of the achievement on which the award is based.

2. The presentation ceremonies will take place preferably during the Annual Meeting of the Institute, although the Directors may arrange for its presentation elsewhere.

3. The medal will be awarded for distinguished achievement in mining. There are no limitations as to nationality, membership in our Institute or otherwise.

The recipient must be a living person, able and willing to present himself in person to receive the award at the time and place prescribed by the Board of Directors.

4. The choice of a medallist is to be based upon a report by a Mining Medal Committee of fifteen members of the Institute interested in mining, chosen by the Board of Directors on the nomination of the President, and distributed as widely geographically as convenient. The tenure of office shall be for three years, so arranged that a portion of the Committee shall be elected each year. A member shall not be eligible for reappointment to the Committee for a period of one year after his term has expired. The President of the Institute shall, ex officio, also be a member of the Committee, but not its chairman.

The award shall finally be made by a majority vote of the Board of Directors of the Institute after consideration of the report of the Committee. The Directors shall have the right to reject the report of the Committee.

5. The medal committee may invite nominations from the membership of the Institute, such nominations to be accompanied by an argument in favor of the award and details sufficient for a proper citation to be drawn.

This medal has been awarded as follows:

1927—David William Brunton.
1928—Herbert Hoover.
1929—John Hays Hammond.
1930—Daniel C. Jackling.

Miscellaneous Prizes

Many of the Local Sections have established or administer prizes for the best paper presented within the year by any junior member.

Annual Lectures

Howe Memorial Lecture

The Howe Memorial Lecture, in memory of Henry Marion Howe, Past President of the Institute, was authorized in April, 1923, as an annual address to be delivered by invitation under the auspices of the Institute by an individual of recognized and outstanding attainment in the science and practice of iron and steel metallurgy or metallography, chosen by the Board of Directors upon recommendation of the Iron and Steel Division. The following were the Howe lectures and lecturers for the years indicated:

1924 What is Steel? By Albert Sauveur.
1925 Austenite and Austenitic Steels. By John A. Mathews.
1926 Twenty-five Years of Metallography. By William Campbell.
1927 Alloy Steels. By Bradley Stoughton.
1928 Significance of the Simple Steel Analysis. By Harry D. Hibbard.
1929 Studies of Hadfield's Manganese Steel with the High-power Microscope. By John Howe Hall.
1930 The Future of the American Iron and Steel Industry. By Zay Jeffries.

Institute of Metals Lecture

An annual lectureship was established in 1921 by the Institute of Metals and it has come to be one of the important functions of the Annual Meeting. A number of distinguished men from this country and abroad have served in this lectureship. The roll is quoted below:

1922 Colloid Chemistry and Metallurgy. By Wilder D. Bancroft.
1923 Solid Solution. By Walter Rosenhain.
1924 The Trend in the Science of Metals. By Zay Jeffries.
1925 Action of Hot Wall: a Factor of Fundamental Influence on the Rapid Corrosion of Water Tubes and Related to the Segregation in Hot Metals. By Carl Benedicks.
1926 The Relation between Metallurgy and Atomic Structure. By Paul D. Foote.
1927 Growth of Metallic Crystals. By Cecil H. Desch.
1928 Twining in Metals. By C. H. Mathewson.
1929 The Passivity of Metals, and Its Relation to Problems of Corrosion. By Ulick R. Evans.
1930 Hard Metal Carbides and Cemented Tungsten Carbide. By S. L. Hoyt.

Joint Activities

The Institute conducts jointly with the American Society of Civil Engineers, American Society of Mechanical Engineers and American Institute of Electrical Engineers, certain activities as listed below, and is joint owner with them of certain properties.

United Engineering Trustees, Inc.*

When Mr. Carnegie made his gift which made possible a home for the engineering societies, a separate corporation was organized to act as a holding company and to manage the common property. This was long known as the United Engineering Society but the name has now been changed to one which more nearly reflects the real function of the corporation. It holds title to the Engineering Societies Building and to various funds placed in its charge and is managed by a Board of Trustees of which three are chosen by each of the four societies, American Institute of Mining and Metallurgical Engineers, American Society of Mechanical Engineers, American Institute of Electrical Engineers and American Society of Civil Engineers, which own equal and undivided interests in the property. The land and building cost two million dollars. The interest of each of the Founders societies, aside from especial trust funds, is now valued at $493,352. The A. I. M. E. trustees are Arthur S. Dwight, J. V. N. Dorr and H. A. Guess. George D. Barron will succeed Mr. Guess as trustee at the end of this year. The President of the Board is Francis Lee Stuart and the Secretary, Alfred D. Flinn.

The Engineering Foundation

The Engineering Foundation was established in 1914 as a result of a gift of $200,000 made by Ambrose Swasey. It is intended not only "for the furtherance of research" along broad lines but also "for the advancement of" a wide variety of other activities of benefit to engineering as need arises. The endowment fund, which now amounts to $630,000, is administered by the United Engineering Trustees, Inc. Through The Engineering Foundation board, on which the Institute is now represented by J. V. N. Dorr, John M. Callow and Galen H. Clevenger, the income of the endowment, approximately $28,000 a year, and funds contributed for special projects are appropriated for the purposes intended. H. C. Bellinger will succeed Mr. Callow on the board at the end of this year. Alfred D. Flinn is Secretary and Director.

Researches of wide interest in many lines have been aided by Engineering Foundation. A grant from it to the Mining Methods Committee assisted in the preparation of Volume 72 of the TRANSACTIONS. For several years the Foundation has aided in support of the study of Blast-furnace Slags by R. S. McCaffery at the University of Wisconsin, sponsored by the Institute. The Foundation has raised over $230,000 to support the Alloys of Iron Research endorsed by the Institute and George B. Waterhouse is serving as Chairman and Director of the Iron Alloys Committee which has been brought together to direct this work. The Foundation has also on recommendation of the Institute made a grant in aid of the studies of Mining and Strata being conducted at Columbia University by P. B. Bucky, which promise information of great usefulness relative to mine subsidence and support of ground.

John Fritz Medal Board of Award

The medal was established in 1902 and has since been awarded not oftener than annually by a board of sixteen members, four representatives each from the American Societies of Civil, Mining and Metallurgical, Mechanical, and Electrical Engineers.

* New name in process of legal adoption at end of year.

This medal is awarded "for notable scientific or industrial achievement," and there is no restriction on account of nationality or sex. The board meets annually on the third Friday of October. For 1930–31, the President is Lincoln Bush; Honorary Treasurer, H. B. Smith; Honorary Secretary, Alexander Dow; Assistant Secretary, Alfred D. Flinn, 29 West 39th St., New York.

Alfred Noble Prize

The American Society of Civil Engineers acts as custodian of a fund from which the income provides a prize of $500 awardable to members of that society, the American Society of Mechanical Engineers, the American Institute of Electrical Engineers, the American Institute of Mining and Metallurgical Engineers and the Western Society of Engineers. The terms of the award provide that the prize may be given "for a technical paper of particular merit accepted by any of the foregoing societies for publication, in whole or in abstract, in any of their respective technical publications, provided that the author, at the time the paper is accepted in practically its final form by the proper committee of the society publishing it, be not over thirty years of age." J. Vipond Davies represents the Institute on the Committee in Charge.

Hoover Medal Board of Award

This medal was established in 1929 to be awarded for Distinguished Public Service by an Engineer and the first award was made to Herbert Hoover. The funds were placed in the custody of the American Society of Mechanical Engineers but the award is made by a joint committee of representatives of that society and of the American Institute of Electrical Engineers, American Society of Civil Engineers and American Institute of Mining and Metallurgical Engineers. On this board J. V. W. Reynders, Richard Peters, Jr. and Charles V. Drew represent the American Institute of Mining and Metallurgical Engineers.

Joint Conference Committee

This is a Committee consisting of the presidents and secretaries of the four National Engineering Societies, which meets at intervals to consider matters of mutual concern to the four societies and makes, where possible, recommendations for concurrent action by the societies.

Engineering Societies Library

This is a consolidation of the four libraries formerly maintained by the four national engineering societies and is administered through the Engineering Societies Library Board. It contains over 135,000 volumes and is conducted as a free public reference library. It is open to the public from 9 a.m. to 10 p.m. on week days, excepting July and August, when the library closes at 5 p.m. A. I. M. E. representatives for 1930 were: Sydney H. Ball, George C. Stone, Philip W. Henry and Joseph E. Pogue. HARRISON W. CRAVER, Director, 29 West 39th St., New York, N. Y.

Engineering Societies Employment Service

A cooperative service for engineers and their employers under the direction of the American Institute of Mining and Metallurgical Engineers; American Society of Civil Engineers; American Society of Mechanical Engineers, and American Institute of Electrical Engineers, and with the cooperation in Chicago of the Western Society of Engineers, and in San Francisco of the California Section of the American Chemical Society and the Engineers' Club of San Francisco. For information as to men available or positions open, address the nearest office: New York, Walter V. Brown, Manager, 31 West 39th St., telephone Pennsylvania 9220; Chicago, A. Krauser, Manager, 1216 Engineering Bldg., 205 West Wacker Drive, telephone Harrison 1238; San Francisco, Newton D. Cook, Manager, Room 715, 57 Post St., telephone Sutter 1684.

LIST OF THE MEETINGS OF THE INSTITUTE AND THEIR LOCALITIES FROM ITS ORGANIZATION TO DECEMBER, 1930

No.	Place	Date	Trans. Vol.	Trans. Page
1.	Wilkes-Barre, Pa.*	May, '71	1	3
2.	Bethlehem, Pa.	Aug., '71	1	10
3.	Troy, N. Y.	Nov., '71	1	13
4.	Philadelphia, Pa.	Feb., '72	1	17
5.	New York, N. Y.*	May, '72	1	20
6.	Pittsburgh, Pa.	Oct., '72	1	25
7.	Boston, Mass.	Feb., '73	1	28
8.	Philadelphia, Pa.*	May, '73	2	3
9.	Easton, Pa.	Oct., '73	2	7
10.	New York, N. Y.	Feb., '74	2	11
11.	St. Louis, Mo.*	May, '74	3	3
12.	Hazleton, Pa.	Oct., '74	3	8
13.	New Haven, Conn.	Feb., '75	3	15
14.	Dover, N. J.*	May, '75	4	3
15.	Cleveland, O.	Oct., '75	4	9
16.	Washington, D. C.	Feb., '76	4	18
17.	Philadelphia, Pa.†	June, '76	5	3
18.	Philadelphia, Pa.	Oct., '76	5	19
19.	New York, N. Y.	Feb., '77	5	27
20.	Wilkes-Barre, Pa.*	May, '77	6	3
21.	Amenia, N. Y.	Oct., '77	6	10
22.	Philadelphia, Pa.	Feb., '78	6	18
23.	Chattanooga, Tenn.*	May, '78	7	3
24.	Lake George, N. Y.	Oct., '78	7	103
25.	Baltimore, Md.*	Feb., '79	7	217
26.	Pittsburgh, Pa.	May, '79	8	3
27.	Montreal, Canada	Sept., '79	8	121
28.	New York, N. Y.*	Feb., '80	8	275
29.	Lake Superior, Mich.	Aug., '80	9	1
30.	Philadelphia, Pa.*	Feb., '81	9	275
31.	Staunton, Va.	May, '81	10	1
32.	Harrisburg, Pa.	Oct., '81	10	119
33.	Washington, D. C.*	Feb., '82	10	225
34.	Denver, Colo.	Aug., '82	11	1
35.	Boston, Mass.*	Feb., '83	11	217
36.	Roanoke, Va.	June, '83	12	3
37.	Troy, N. Y.	Oct., '83	12	175
38.	Cincinnati, O.*	Feb., '84	12	447
39.	Chicago, Ill.	May, '84	13	1
40.	Philadelphia, Pa.	Sept., '84	13	285
41.	New York, N. Y.*	Feb., '85	13	585
42.	Chattanooga, Tenn.	May, '85	14	1
43.	Halifax, N. S.	Sept., '85	14	307
44.	Pittsburgh, Pa.*	Feb., '86	14	587
45.	Bethlehem, Pa.	May, '86	15	lxiii.
46.	St. Louis, Mo.	Oct., '86	15	lxx.
47.	Scranton, Pa.*	Feb., '87	15	lxxvi.
48.	Utah and Montana	July, '87	16	xvii.
49.	Duluth, Minn.	July, '87	16	xxiv.
50.	Boston, Mass.*	Feb., '88	16	xxviii.
51.	Birmingham, Ala.	May, '88	17	xix.
52.	Buffalo, N. Y.	Oct., '88	17	xxiv.
53.	New York, N. Y.*	Feb., '89	17	xxxi.
54.	Colorado	June, '89	18	xvii.
55.	Ottawa, Canada	Oct., '89	18	xxiv.
56.	Washington, D. C.*	Feb., '90	18	xxx.
57.	New York, N. Y.	Sept., '90	19	vii.
58.	New York, N. Y.*	Feb., '91	19	xxv.
59.	Cleveland, O.	June, '91	20	xvi.
60.	Glen Summit, Pa.	Oct., '91	20	lxi.
61.	Baltimore, Md.*	Feb., '92	21	xix.
62.	Plattsburg, N. Y.	June, '92	21	xxxiii.
63.	Reading, Pa.	Oct., '92	21	xliv.
64.	Montreal, Canada*	Feb., '93	21	lii.
65.	Chicago, Ill.	Aug., '93	22	xiii.
66.	Virginia Beach, Va.*	Feb., '94	24	xvii.
67.	Bridgeport, Conn.	Oct., '94	24	xxxv.
68.	Florida‡	Mar., '95	25	xix.
69.	Atlanta, Ga.	Oct., '95	25	xxxiii.
70.	Pittsburgh, Pa.*	Feb., '96	26	xvii.
71.	Colorado	Sept., '96	26	xxix.
72.	Chicago, Ill.	Feb., '97	27	xvii.
73.	Lake Superior	July, '97	27	xxx.
74.	Atlantic City, N. J.*	Feb., '98	28	xvii.
75.	Buffalo, N. Y.	Oct., '98	28	xxxvi.
76.	New York, N. Y.*	Feb., '99	29	xvii.
77.	California	Sept., '99	29	xlix.
78.	Washington, D. C.*	Feb., '00	30	xix.
79.	Canada	Aug., '00	30	xlv.
80.	Richmond, Va.*	Feb., '01	31	xix.
81.	Mexico	Nov., '01	32	cxviii.
82.	Philadelphia, Pa.§	May, '02	33	xxxv.
83.	New Haven, Conn.	Oct., '02	33	xlvii.
84.	Albany, N. Y.*	Feb., '03	34	xxiii.
85.	New York, N. Y.	Oct., '03	34	lxi.
86.	Atlantic City, N. J.*	Feb., '04	35	xxiii.
87.	Lake Superior	Sept., '04	35	xlii.
88.	Washington, D. C.	May, '05	36	xlii.
89.	British Columbia	July, '05	36	liii.
90.	Bethlehem, Pa.	Feb., '06	37	xli.
91.	London, England	July, '06	37	xlviii.
92.	New York, N. Y.	April, '07	38	lii.
93.	Toronto, Canada	July, '07	38	lix.
94.	New York, N. Y.	Feb., '08	39	xli.
95.	Chattanooga, Tenn.	Oct., '08	39	xlviii.
96.	New Haven, Conn.	Feb., '09	40	xli.
97.	Spokane, Wash.	Sept., '09	40	xlviii.
98.	Pittsburgh, Pa.	Mar., '10	41	xxxviii.
99.	Canal Zone	Nov., '10	41	xlv.
100.	Wilkes-Barre, Pa.	June, '11	42	xxxiv.
101.	San Francisco, Cal.	Oct., '11	42	xliv.
102.	New York, N. Y.*	Feb., '12	43	lxxvii.
103.	Cleveland, Ohio	Oct., '12	44	vii.
104.	New York, N. Y.*	Feb., '13	45	xv.
105.	Butte, Mont.	Aug., '13	46	vii.
106.	New York, N. Y.	Oct., '13	47	vii.
107.	New York, N. Y.*	Feb., '14	48	xv.
108.	Salt Lake City, Utah	Aug., '14	49	vii.
109.	Pittsburgh, Pa.	Oct., '14	50	vii.
110.	New York, N. Y.*	Feb., '15	51	xvi.
111.	San Francisco, Cal.	Sept., '15	52	vii.
112.	New York, N. Y.*	Feb., '16	54	xvi.
113.	Arizona	Sept., '16	55	vii.
114.	New York, N. Y.*	Feb., '17	56	vii.
115.	St. Louis, Mo.	Oct., '17	57	vii.
116.	New York, N. Y.*	Feb., '18	59	xvii.
117.	Colorado	Sept., '18	60	vii.
118.	Milwaukee, Wis.	Oct., '18	60	xxii.
119.	New York, N. Y.*	Feb., '19	61	xi.
120.	Chicago	Sept., '19	62	vii.
121.	New York, N. Y.*	Feb., '20	63	ix.
122.	Lake Superior District	Aug., '20	66	ix.
123.	New York, N. Y.*	Feb., '21	66	xiii.
124.	Wilkes-Barre, Pa.	Sept., '21	66	xix.
125.	New York, N. Y.*	Feb., '22	67	ix.
126.	San Francisco, Cal.	Sept., '22	68	xxix.
127.	New York, N. Y.*	Feb., '23	69	xxix.
128.	Canada	Aug., '23	69	xxxvi.
129.	New York, N. Y.*	Feb., '24	70	xxxi.
130.	Birmingham, Ala.	Oct., '24	71	xxxiii.
131.	New York, N. Y.*	Feb., '25	71	xxxviii.
132.	Salt Lake City, Utah	Sept., '25	73	xvi.
133.	New York, N. Y.*	Feb., '26	73	xxii.
134.	Pittsburgh, Pa.	Oct., '26	74	xv.
135.	New York, N. Y.*	Feb., '27	75	xvii.
136.	New York, N. Y.*	Feb., '28	76	104‖
137.	New York, N. Y.*	Feb., '29	29¶	116‖
138.	San Francisco, Cal.	Oct., '29	30¶	496‖
139.	New York, N. Y.*	Feb., '30	30¶	132‖

* Annual meeting for the election of officers. The rules were amended at the Chattanooga meeting May, 1878, changing the annual election from May to February.

† Begun in May at Easton, Pa., for the election of officers, and adjourned to Philadelphia.

‡ Begun in February at New York City, for the election of officers, and adjourned to Florida.

§ Begun in February at New York City, for the election of officers, and adjourned to Philadelphia.

‖ See Mining and Metallurgy of year of meeting on page indicated for complete story of meeting.

¶ See Transactions for year indicated.

New York Meeting

The 139th meeting* of the American Institute of Mining and Metallurgical Engineers was held at New York, February 17 to 20, 1930, the attendance totaling approximately 1800. The meeting consisted of the annual business session, 34 technical sessions at which were presented 166 papers, two meetings of the Board of Directors, two meetings of Section Delegates, two formal lectures and one special lecture, five group dinners and one group luncheon, luncheon daily for members and guests, fourteen committee meetings and conferences, the President's reception, the dinner-dance, an informal dance, a smoker, an exhibition of rare minerals and metals, two meetings of the Woman's Auxiliary and a special program of entertainment for the ladies.

The Petroleum Division held seven sessions, including two general sessions featuring, respectively, a symposium on unit operation of oil pools and reports on developments in the petroleum industry during the previous year. The Iron and Steel Division held four sessions, at one of which Dr. Zay Jeffries delivered the Howe Memorial Lecture, the title of which was "The Future of the American Iron and Steel Industry." The Institute of Metals Division held six sessions, including two devoted to a symposium on the Melting and Casting of Metals, and the general session at which Dr. S. L. Hoyt delivered the annual Institute of Metals Lecture. The title of his lecture was "Hard Metal Carbides and Cemented Tungsten Carbide."

The Coal and Coal Products Committee devoted one session and the greater part of another to Coal Classification and at a third session organized as the Coal Division of the Institute. The Committees on Mining Methods and Ground Movement and Subsidence each held a session, besides which they met jointly. Sessions were held by other Technical Committees as follows: Nonferrous Metallurgy, 1; Geophysical Prospecting, 2; Nonmetallic Minerals, 2; Milling Methods, 1; Rare Minerals and Metals, 1; Mining Geology, 2; Mine Ventilation, 2; Engineering Education, 1.

At the annual business meeting on February 18, the following ticket was elected, and the reports of the President, Treasurer and Secretary were presented:

William H. Bassett, President; Scott Turner, Vice-president; Henry Krumb, Vice-president; R. C. Allen, Cadwallader Evans, Jr., John M. Lovejoy, John A. Mathews and Milnor Roberts, Directors.

At an executive session of the Directors on Tuesday afternoon, H. A. Guess was elected Vice-president to fill the vacancy created by the

* For program of meeting see MINING AND METALLURGY (Feb., 1930) 76; for news story see the March, 1930, number.

elevation of Mr. Bassett to the presidency; Karl Eilers was elected to succeed himself as Treasurer, and H. Foster Bain, as Secretary.

Representatives of 23 Sections and three Divisions attended the meeting of Section Delegates on Monday morning, the most complete representation as yet brought together in this way.

The Woman's Auxiliary held its annual meeting Tuesday morning and afternoon. In all 216 were present. Teas, dances, a theater party, luncheons, excursions and many other interesting activities filled the four days to overflowing for the ladies.

About 500 members and guests attended the informal dance on Tuesday night in the Engineering Societies Building.

The Annual Smoker was a very successful affair, more than 500 men turning out, 100 more than the previous year.

The Coal and Coal Products Committee held a luncheon at the Engineers' Club, at which the speakers were C. E. Bockus, president of the National Coal Association, and Theodore Marvin, editor of *The Explosives Engineer*. G. H. Clamer, president of the Ajax Metal Co., addressed those who attended the annual dinner of the Institute of Metals Division at the Savoy-Plaza Hotel on Thursday evening. The annual dinner of the Petroleum Division was held at the Engineers' Club, also on Thursday evening

The annual dinner-dance was held at the Commodore Hotel, with 1000 in attendance, the next to the largest attendance at an Institute dinner. The incoming and retiring Presidents held a reception preceding the dinner. Ralph M. Roosevelt, chairman of the New York Section, was toastmaster. The members of the Class of 1880 of the Institute Legion of Honor were introduced, George E. Thackray responding for the group. The James Douglas gold medal was presented to John Van Nostrand Dorr. The Robert W. Hunt medal was presented to James Aston. William S. Unger was the recipient of the Joseph E. Johnson, Jr., prize. Retiring President Frederick W. Bradley presented his presidential address.

Pittsburgh Meeting

The first fall meeting* of the new Coal Division was held at the William Penn Hotel, Pittsburgh, Pa., September 11, 12 and 13. The total registration was 248. More than 20 papers were presented and discussed, including a series of papers by the Junior Section of the Division, which was organized in the spring. Two of the four sessions were devoted to anthracite and bituminous coal preparation, respectively.

One hundred members and guests attended the Division dinner, over which M. D. Cooper presided as toastmaster. The speakers were: Chairman H. N. Eavenson, Past President S. A. Taylor, Major K. C. Appleyard, F. Prockat and K. S. Twitchell.

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* For news story of meeting see Mining and Metallurgy (Oct., 1930) 483.

On the last day of the meeting members and visitors were organized into four groups to visit Hazelwood, the Champion mine, Neville Island and the Colfax plant of the Duquesne Light Co., respectively. At Hazelwood, coal washing at the rate of 100 tons per hour in Simon-Carves machines and by-product coking in the 360 ovens of the Jones & Laughlin Steel Corpn. were shown the visitors. At the Champion mine, interest centered in the Rheolaveur plant of the Pittsburgh Coal Co. On Neville Island the Davison Coke & Iron Co. had many interesting things to exhibit. At the Colfax power station use of pulverized coal on a large scale was seen and at the Springdale plant of the West Penn Electric Co., also visited, the slagging type of furnace was examined with interest.

Chicago Meeting

The Institute of Metals and Iron and Steel Divisions met* jointly at the Stevens Hotel, Chicago, during the week of the National Metal Congress, September 22 to 26. The Institute of Metals Division held four sessions, one general in nature, one devoted to Alloys and one to Aluminum, and a joint session with the Iron and Steel Division on Theoretical Metallurgy. The latter Division held also a session and a round table on Iron Ore.

The only social function was a joint dinner of the two Divisions on Tuesday evening. W. J. MacKenzie, chairman of the Iron and Steel Division, acted as toastmaster, and brief speeches were made by President W. H. Bassett, G. E. Johnson, chairman of the Chicago Section; Dr. Zay Jeffries, chairman, Institute of Metals Division; A. B. Kinzel, secretary, Iron and Steel Division; R. G. Guthrie, president of the American Society for Steel Treating, and J. R. Van Pelt, secretary, Chicago Section. B. D. Saklatwalla, vice-president of the Vanadium Corpn. of America, gave a semitechnical talk on vanadium.

The Executive Committee of the Institute of Metals Division held a luncheon meeting on Tuesday at which plans for the future were discussed. Dr. Jeffries presided. A similar meeting was held on Wednesday by the Iron and Steel Division with Mr. MacKenzie presiding.

Two trips were available to members of the Iron and Steel Divisions—to the South Works of the Illinois Steel Co. and the 118th Street Steel Works of the Republic Steel Corpn., respectively. Those interested in nonferrous metals had the choice of trips to two plants, the Hawthorne plant of the Western Electric Co. and the East Chicago refinery of the International Lead Refining Co.

Tulsa Meeting

The Petroleum Division held its first fall meeting† at Tulsa, October 2 and 3, preceding the International Petroleum Exposition (October 4 to

* For news story of meeting see MINING AND METALLURGY (Nov., 1930) 509.

† For news story see MINING AND METALLURGY (Nov., 1930) 515.

11). The Hotel Mayo was headquarters; total registration, 215. The first day of the meeting was devoted to papers on the unit operation of oil pools, the second day to papers on production engineering subjects. The discussion was almost continuous, with barely time out for meals. On the first day, W. P. German, J. A. Veasey and Henry L. Doherty discussed the legal situation. Mr. German had just represented the Mid-Continent Oil and Gas Association in litigation in which the Federal Court had sustained the power of the State Corporation Commission to enforce proration orders. Luncheon was served in the same room where the sessions were held, 70 to 80 lunching together each day.

One hundred attended the informal dinner at the Tulsa Club Thursday evening.

El Paso Meeting

A regional meeting* of the Institute was held at El Paso, Texas, October 13–16, with the Hotel Paso del Norte as headquarters. It was held jointly with the Western Division of the American Mining Congress and the West Texas Geological Society, the Centro Nacional de Ingenieros and other organizations cooperated in the program. Nine papers were presented and discussed at the A. I. M. E. sessions on Tuesday, October 14, at which approximately 100 were present.

The Board of Directors held its regular monthly meeting at the Paso del Norte on Tuesday evening, the local committee entertaining the officers and directors of the Institute at the dinner preceding the meeting. Speakers at the dinner were President W. H. Bassett, C. V. Millikan, chairman of the Petroleum Division; Milnor Roberts, E. P. Mathewson and Eugene McAuliffe.

The entertainment included a visit to Juarez, Mexico, where a barbecue supper was served, various luncheons and excursions and an elaborate banquet at Hotel del Norte on the final evening.

Los Angeles Meeting

The second fall meeting† of the Petroleum Division was held at Los Angeles, Calif., October 17, and more than 200 in attendance made it necessary to transfer the meeting from the place originally selected to the Biltmore Hotel. A dozen or more papers were presented and discussed at the three sessions.

The dinner in the evening was presided over by C. V. Millikan, chairman of the Division. Two hundred and thirty-one were present.

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* For preliminary and news stories of meeting see Mining and Metallurgy (Oct., 1930) 467 and (Nov., 1930) 512.

† For news story of meeting see Mining and Metallurgy (Nov., 1930) 517.

Official Institute Reports for the Year 1929

The official Institute reports for the year 1929 were distributed in pamphlet form at the Annual Meeting, February, 1930, and were later included in Section 2 of MINING AND METALLURGY, June, 1930, and thus made available to the entire membership.

Membership

In its report the Admissions Committee set forth the changes in membership during 1929, as follows: Total membership, Jan. 1, 1928, 8582; elected and placed on rolls in 1929, including 59 reinstatements, 509; loss by resignations in 1929, 88; loss by death in 1929, 110; loss by suspension in 1929, 37; membership at end of year not including Student Associates, 8856. Distribution of membership: Honorary Members, 16; Members, 6590; Junior Members, 38; Associates, 1041; Junior Associates, 1033; Rocky Mountain, 138; total membership, Jan. 1, 1930, 8856; Student Associates, 150; total including student associates, 9006.

Treasurer's Annual Report, Year of 1929

STATEMENT OF RECEIPTS

For the year ended December 31, 1929

Dues:		
Arrears	$ 3,501.75	
Current	97,257.12	
New Members	5,721.50	
Advance	1,970.79	$108,451.16
Initiation Fees		7,436.00
Magazine Receipts:		
Advertising Gross Receipts	$40,610.81	
Magazine Sales	3,799.84	44,410.65
Sale of Transactions Current	$ 2,985.15	
Sale of Special Editions Current	5,147.82	
Sale of Back Transactions and Special Editions	4,186.93	
Sale of Year Book	55.50	
Sale of Authors' Reprints	2,585.40	
Sale of Pins and Fobs	14.73	
Sale of Technical Publications	1,451.55	
Sale of Bindings	3,949.13	20,376.21
Net Interest Received:		
On Bank Balance and Temporary Investments	$ 2,595.04	
On Life Membership Fund	3,223.43	
On James Douglas Library Fund	5,425.61	
On General Reserve Fund	2,015.58	13,259.66
Miscellaneous Receipts		335.80
Total Receipts		$194,269.48

Statement of Expenditures

For the year ended December 31, 1929

Magazine Expenditures			$ 41,043.19
All Other Publication Expense:			
Transactions:			
Volume 1 to 76, inclusive	$ 1,228.15		
General Volume 1929	13,192.40		
General Volume 1930	764.41	$15,184.96	
Technical Publications		11,972.82	
Petroleum Division—Volume 1929		5,983.62	
Petroleum Division—Volume 1930		482.18	
Institute of Metals Division—Volume 1929		5,113.69	
Institute of Metals Division—Volume 1930		769.92	
Milling and Concentration—Volume 1930		1,230.89	
Coal and Coke—Volume 1930		717.91	
Geophysical Volume—1929		3,107.61	
Iron and Steel Volume—1929		4,852.63	
Authors' Reprints		1,515.24	
Year Book		3,480.46	
Special Editions		471.76	54,883.69
Joint Activities:			
Library Assessment		$ 7,287.60	
Employment Service		424.02	
American Standards Association		1,000.00	
John Fritz Medal Board of Award		1,404.22	10,115.84
Meeting—Technical Committees, Local Sections:			
Local Section Appropriations		$ 2,975.00	
Meetings		8,922.32	
Local Section Traveling Expense		2,514.28	
Technical Committees and Divisions		934.54	15,346.14
Accounting, Membership and Membership Increase Departments:			
Accounting Department Salaries and Expense		$ 8,589.00	
Membership Department Salaries and Expenses		2,367.63	
Increase of Membership Department Salaries and Expense		5,892.89	16,849.52
General Office Salaries			35,277.04
Miscellaneous General Office Expense:			
Office Space		$ 1,944.00	
Stationery and Supplies		1,488.00	
Telephone and Telegraph		1,676.56	
Postage (General Office Mail)		297.04	
Furniture and Fixtures (Including $1,500.00 Depreciation)		1,979.06	
Miscellaneous Expenses		974.36	
Traveling Expenses of President and Secretary		2,326.91	
Traveling Expenses of Directors		413.35	
Legal Expenses		250.00	
Insurance		19.33	11,368.61
Contingencies			1,231.50
Total Expenditures			$186,115.53

Assets

Founders Interest in Real Estate and Other Assets of United Engineering Society, except Trust Funds			$491,642.36
Books in Library			40,000.00
Investments:			
Life Membership Fund		59,278.45	
James Douglas Library Fund		100,103.50	
Robert W. Hunt Fund			
Principal	7,502.25		
Income	782.25	8,284.50	
J. E. Johnson, Jr. Fund		2,922.63	
James Douglas Medal Fund		3,270.00	
General Reserve Fund		36,665.12	
William Lawrence Saunders Mining Medal Fund....		7,000.00	
Anthony F. Lucas Fund			
Principal	5,482.00		
Income	447.00	5,929.00	
Rocky Mountain Club Fund			
Principal	109,518.75		
Income	4,525.75	114,044.50	
Seeley W. Mudd Memorial Fund			
Principal	99,958.75		
Income	2,570.25	102,529.00	
Charles F. Rand Memorial Medal Fund		10,148.25	
		450,174.95	
Reserve for Depreciation of Furniture and Fixtures...		1,500.00	
Temporary Investment		8,685.00	
Total			460,359.95
Cash			19,906.12
Accounts Receivable			1,442.10
Equipment, Furniture and Fixtures		14,317.37	
Less—Reserve for Depreciation		3,035.31	11,282.06
Inventories:			
Paper on hand		641.11	
Books—Transactions and Special Editions		20,839.85	
Postage on hand		742.40	
Supplies		918.83	
Total Inventories			23,142.19
			$1,047,774.78

Liabilities

Accounts Payable			$ 14,195.27
Due the Mining Methods Committee			653.38
Life Membership Fund		59,733.92	
James Douglas Library Fund		100,000.00	
Robert W. Hunt Fund			
Principal	$ 7,500.00		
Income	702.26	8,202.26	
J. E. Johnson, Jr. Fund			
Principal	3,000.00		
Income	343.07	3,343.07	
James Douglas Medal Fund			
Principal	3,250.00		
Deficit	110.37	3,139.63	
William Lawrence Saunders Medal Fund			
Principal	7,000.00		
Income	217.74	7,217.74	
Anthony F. Lucas Fund			
Principal	5,490.00		
Income	544.95	6,034.95	
Rocky Mountain Club Fund			
Principal	109,533.72		
Income	4,673.21	114,206.93	
General Reserve Fund		36,656.35	
Seeley W. Mudd Memorial Fund			
Principal	100,000.00		
Income	2,657.66	102,657.66	
Charles F. Rand Memorial Medal Fund			
Principal	10,196.06		
Deficit	233.30	9,962.76	
Total due the Funds			451,155.27
Reserve for Life Memberships			40,000.00
Surplus, as follows:			
Contributions by Andrew Carnegie and Others to Real Estate		491,642.36	
Consisting in part of Equipment Furniture and Fixtures and Book Inventories		50,128.50	541,770.86
			$1,047,774.78

Karl Eilers,
Treasurer.

We hereby certify that we have audited the books and account of the American Institute of Mining and Metallurgical Engineers and that the above statement of Assets and Liabilities is in accord therewith as at December 31, 1929.

Loomis, Suffern, Fernald,
Certified Public Accountants.

January 14, 1930.

Necrology

The following is a list of members who died in 1929. It is compiled from reports to the Secretary's office. Biographical sketches published in Mining and Metallurgy are indicated in the last two columns.

Year of Election	Name	Date of Death	Issue Containing Biography 1929	Page
1922	Adams, Waldo Peck	June 2	September	435
1903	Argall, Joseph	Nov. 1		
1894	Austin, Leonard S.	Oct. 30	December	580
1905	Bailey, Melbourne	Dec. 12	May*	280
1890	Barringer, Daniel M.	Nov. 30	February*	119
1895	Batcheller, Henry R.	Jan. 15	March	167
	Bingham,† Charles E.			
1923	Birkett, E. H.	Apr. 23	August	394
1913	Bissell, Robert W.	Jan. 18	April	214
1926	Blackmar, Don C.	June 4	March*	189
1920	Bosworth, T. O.	Jan. 18	April	214
1894	Boucher, Arthur S.	Jan. 6	May	257
1927	Brandenthaler, R. R.	Dec. 14	January*	64
1889	Bretherton, S. E.	Oct. 4	December	581
1911	Brunton, F. K.	Aug. 17		
1924	Brush, Charles F.	June 15	September	436
1925	Cameron, Frank N.	May 16	July	349
1891	Chambers, Robert E.	May	July	348
1908	Clark, J. Murray	Dec. 3	August*	407
1883	Clymer, Edward T.	Oct. 13	December	579
1914	Coffey, George T.	May 8	July	349
1901	Cooke, L. H.	Aug. 23	November	537
1915	Cousins, George W.	June 30	February*	118
1886	Cowles, A. H.	Aug. 13	September	436
1892	Crane, Theron I.	Nov. 2	December	579
1921	Crowell, Samuel B.	Dec. 9	January*	64
1907	Curtis, Bracey	Jan. 31	April	215
1895	de Batz, Rene	January	March*	189
1928	Dines, Tyson S.	March	May	257
1909	Dufourcq, R. G.	January	November	537
1920	Ericsson, Eric A.	Apr. 4	April*	231
1920	Erlenborn, Willi	Nov. 10	March*	189
1888	Farish, John B.	Nov. 14	December	580
1891	Ferguson, Vincent	Mar. 8	May	257
1917	Fincane, T. R.	Mar. 22	February*	118
1921	Foch, Ferdinand	Mar. 20		
1885	Foucar, Edouard L.	Jan. 3	March	167
1880	Francklyn, Charles G.	Jan. 11	April	214
1925	Friedl, Arthur J.	July 9	September	435
1894	Fuller, James W. Jr.	April 4		
1876	Goodale, Charles W.	April 11	May	258
1912	Graybill, John	Dec. 11	June*	320
1887	Gritzner, F. A.	Mar. 26	July	394
1906	Gunther, C. Godfrey	Dec. 26	February*	118
1900	Haggen, Edward A.	Apr. 22	October	488
1895	Hall, Benjamin M.	Nov. 19	March*	189
1913	Harper, Walter S.	Feb. 15	August	394
1921	Hawley, William S.	Apr. 26	June	298
1920	Herd, Walter	Aug. 23	September	435
1889	Holt, Marmaduke B.	Dec. 26	February*	118
1928	Husk, Albert D.	June 9	December	580
1920	Hutchinson, S. Pemberton	Feb. 16	April	214
1926	Ivanoff, Constantine L.	Mar. 7	June	298
1914	Jaeger, Frederick	July 28	October	488
1891	Jennings, Robert E.	Oct. 30	December	580
1922	Johnson, Homar L.	Sept. 17	October	483
1927	Johnson, Homer H.	Mar. 21	May	257
1918	Jones, Jesse L.	Jan. 22	April	214
1892	Jones, Thomas J.	June 18	October	488
1883	Joüet, Cavalier H.	Mar. 29	May	258
1881	Kennedy, John S.	Apr. 22	August	394
1881	Kennedy, Hugh	May 23	September	435
1895	Kervin, James H.	September	October	488
1926	Krom, L. J.	Dec. 24	January*	64
1905	La Follette, Harvey M.	Sept. 19	November	538
1920	MacDonald, Alexander	May	August	394
1904	Marengo, Paolo		May	257
1925	Maxwell, Allison R.	May 7	November	538

* 1930 Issue.
† A Rocky Mountain Club member of the Institute.

Year of Election	Name	Date of Death	Issue Containing Biography 1929	Page
1916	McGrath, Ray P.	Aug. 25	October	488
1890	Mills, Charles E.	Jan. 19	July	350
1915	Montgomery, John C.	Sept. 1	December	581
1926	Mouser, Otis	Nov. 7	December	580
1882	Murray, George			
1918	North, W. O.	Jan. 17	March June	167 298
1915	Notman, George	Dec. 11	January*	64
1884	Noyes, William S.	Oct. 17	March*	189
	O'Connor,† John W	Aug. 5		
1874	Olcott, E. E.	June 5	July	350
1906	Paine, William A.	Sept. 24	February*	117
1911	Peck, Walter R.	Oct. 10	February*	118
1920	Pollok, R. R.	June 30	September	435
1903	Poole, William	July 16	September	435
1911	Quin, Robert A.	Jan. 22	April	213
1891	Randolph, Beverley S.	Feb. 5	June	298
1912	Raymer, George S.	June	July	349
1919	Reinbold, Herman	Nov. 2	January*	64
	Richardson, W. P.	May 20	August	395
1916	Riser, George C. Jr.	Feb. 25	May	258
1913	Risque, James B.	July 12	October	488
1914	Robertson, William A.	Jan. 17	May	257
1881	Robertson, William Fleet	June 11	August	395
1918	Ross, Henry A.	May 10	August	395
1921	Ross, William G.	April	February*	118
1898	Ruhl, J. Louis A.	June 27	August	394
1892	Runyon, Walter C.	Feb. 6	December	579
1908	Schmidt, Henry C.		June	297
1915	Seagrave, W. H.	Feb. 5	April	215
1924	Snyder, Harry	Oct. 11	December	581
1882	Sperr, Frederick W.	Nov. 19	January*	64
1902	Staver, W. H.	Jan. 1	February	96
1899	Thompson, William			
1915	Tickner, Frank W.	April	September	538
1925	Tonkin, John H.	May	July	349
1918	Tyson, George E.	March	February*	118
1897	Van Ness, William W.	Sept. 2		
1909	Wagner, Percy A.	Nov. 11	April*	231
1888	Waldo, Leonard	Jan. 25	April	215
1923	Wardle, Charles W.	Jan. 10	April	214
1919	Wetzel, Thomas A.	January	July	349
1906	White, Edward A.	Nov. 14	February*	118
1928	White, James A.	December	January* October*	64 496
1905	Zwicky, William E.	Oct. 26	December February*	581 118

† A Rocky Mountain Club member of the Institute.

Miami Copper Company Method of Mining Low-grade Orebody

By F. W. Maclennan,* Miami, Ariz.

(New York Meeting, February, 1930)

Ore production from the property of the Miami Copper Co. began early in 1911. Until 1925 this ore came from the so-called high-grade orebodies, which contained a little over 2 per cent. copper. This ore was mined by the following methods:

1. Top slicing after mining the peaks by square setting, which produced 4,524,347 tons, including ore from square setting.
2. Shrinkage stoping with sublevel caving of the pillars, which produced 2,230,577 tons.
3. Undercut caving with hand tramming, which produced 15,427,672 tons.

Estimating December tonnage, the total ore mined to the end of 1929 will amount to slightly more than 43,000,000 tons.

The first method was described by E. G. Deane[1] and the second by D. B. Scott,[1] and the third, together with a general description of the mine, by J. H. Hensley, Jr.[1] The geology of the district has been worked out by Dr. F. L. Ransome.[2]

The author acknowledges also the assistance of R. W. Hughes, assistant mine superintendent, E. V. Graybeal, chief mine engineer, A. J. McDermid and S. R. Burdick, engineers of Miami Copper Co., Miami, Arizona.

Factors Influencing Selection of Mining Methods

The mining method described herein was selected primarily with a low mining cost in view and was developed to avoid as far as possible dilution of the already very low-grade ore.

In 1924, with the exhaustion of the high-grade orebodies in sight, it was decided to develop the low-grade orebody, which at that time contained 36,000,000 tons of ore assaying 1.06 per cent. copper, and had an area of 50 acres with an average thickness of 206 ft. overlain by an average thickness of 320 ft. of barren capping.

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* General Manager, Miami Copper Co.

[1] D. B. Scott: Stoping Methods of Miami Copper Co. *Trans.* A. I. M. E. (1916) **55**, 137.

E. G. Deane: Block Method of Top Slicing of the Miami Copper Co. *Idem* (1916) **55**, 240.

J. H. Hensley, Jr.: Mining Methods of the Miami Copper Co. *Idem* (1925) **72**, 78.

[2] F. L. Ransome: The Copper Deposits of Ray and Miami, Arizona. U. S. Geol. Survey *Prof. Paper* 115 (1919).

This orebody had been developed by churn-drill holes from the surface, many of which bottomed in 1 per cent. ore, which was considered the low limit at the time this drilling was done. At this time the mine was served by two haulage levels, one at the 570-ft. level and the other at the 720. The western half of the orebody was adjacent to the existing mine workings and lay above the 570-ft. haulage level; exploration of the ground underneath over a period of a year or more disclosed additional ore on a basis of a low limit of 0.6 per cent. copper. This was figured to be a reasonable basis with the added thickness of ore, providing as it did a back of approximately 300 ft. to be caved in one lift. This ore was accordingly developed for mining from the 720-ft. haulage level.

More recent development in the eastern half of the orebody and under the old high-grade orebody disclosed considerable additional tonnage and a new haulage level is being opened at the 1000-ft. level to serve this ore. As developed to date the tonnage and grade of the low-grade orebody is estimated at 108,461,700 tons assaying 0.88 per cent. copper, of which 0.79 per cent. occurs as sulfide.

The present approximate dimensions of the orebody are: extreme length east and west, 3500 ft.; extreme width north and south, 2700 ft.; area 100 acres; average thickness 325 ft. It is overlain by barren leached capping varying in thickness from 250 to 500 feet.

The mineralization consists of replacement or partial replacement or coating of the primary cupriferous pyrite by chalcocite, usually occurring in the seams and to a lesser extent disseminated through the altered pre-Cambrian schist. This rock is thoroughly fractured and varies from a hard, highly silicified schist to a soft kaolinized schist, and from a mining point of view may be classified on the average as a free-caving orebody once it had been thoroughly dried by drainage and ventilation.

With an ore calculated to yield approximately 12 lb. net copper per ton, a variation of even 0.1 per cent. in the grade was important, and sampling of the development openings became a major problem. The original churn-drill sampling was checked by (1) the usual channel samples which were cut across the direction of the major seams every 5 ft. of drifts; (2) cuttings from dry stoper-drill holes drilled across the major seams at 2.5-ft. intervals; (3) samples of broken ore taken as the cars were loaded when driving the drifts; (4) a few check samples of 6 to 8 tons of ore shot down from the back of drifts over a length of 25 ft. and carefully quartered down; (5) one 1500-ton sample mined from a narrow shrinkage stope and put through the automatic sampler at the mill. This stope was sampled at the same time by the other methods. The conclusion reached was that the churn-drill samples were accurate, the stoper-drill samples the most accurate small sample for drifts, and the channel sampling average 13 per cent. too high.

Sufficient tonnage was now developed in the low-grade orebody to justify a higher rate of mine production. This was desirable from the

point of view of costs and also in order to maintain a total copper production comparable with that produced in the past from the high-grade orebodies.

Estimates of costs indicated that after deducting all charges exclusive of mining there would remain to cover the cost of mining and to yield a profit approximately 80 c. per ton of ore. It was evident then that low cost should be the outstanding condition affecting the selection of a mining method, and this has been complied with by first intensively developing the orebody in order to make available at the outset its maximum thickness for mining, and, secondly, by developing means of controlling the caving and ore drawing of unusually high backs of ore. A careful detailed estimate was prepared of the cost of mining by the method proposed. Close figuring indicated a possible cost of 38 c. per

FIG. 1.—GENERAL VIEW OF PROPERTY OF MIAMI COPPER CO., LOOKING SOUTHWEST.

ton delivered in the crushing-plant bins. In making up an estimate of total operating costs on which to base an appropriation for the purpose of increasing the operating plant capacity from 6800 tons to 10,000 tons daily, a mining cost of 50 c. per ton was used and it is interesting to note that in mining 16,556,296 tons of ore by this method during the 4-year period from Oct. 1, 1925, when it replaced all other methods, to Oct. 1, 1929, the cost of mining has been 39.94 c. per ton, compared with the estimated figure of 38.

In the earliest caving methods at Miami, the entire width of the orebody was undercut and caved, beginning at one end and retreating along the length of the orebody, closely followed by ore drawing and endeavoring to maintain a plane of contact between this broken ore and barren capping with a dip away from the direction of retreat at an angle of from 40° to 60° from the horizontal. Caving over such a considerable width, 500 or 600 ft., resulted in excessive weight being thrown on the extraction openings, with correspondingly heavy maintenance costs and interference with orderly ore drawing. Later experience indicated that a width of 150 ft. caused satisfactory caving in Miami ore with moderate maintenance costs, and it became standard practice to cave and draw

alternate panels 150 ft. wide across the entire orebody. A year or so later, when the waste rock which had settled into these original panels had consolidated, the pillar panels were caved and drawn back across the orebody between them. This method was well adapted to the comparatively low lifts then in use but it was felt that it would not be satisfactory with the high lifts of 300 ft. or more which were essential to low-cost mining in the low-grade orebody.

In order to maintain an angle of contact between the broken ore and capping of 40° to 60° in an ore column 300 ft. thick, it would be necessary to maintain a length of 300 ft. or more of extraction levels open and in good condition for ore drawing along the panel, and it is likely that this would not always be possible. Excessive weight would almost certainly interfere with orderly ore drawing and in some cases to such an extent that it would be necessary, in order to maintain the required tonnage, to drop back and cave virgin ore, abandoning the broken ore overlying the caved workings. The temptation to do this is always present in the retreating panel system and results in lost ore and increased cost per ton for the reduced tonnage. In this system, employing the inclined plane of contact between broken ore and waste, and with a great thickness of ore, there would be great danger of dilution. There would also be a likelihood that damaging weight would be thrown on the area just ahead of the undercutting by the overhanging cantilever brow of ore which has been undercut but not yet caved. When this happens it is frequently necessary to cut off this cantilever of ore at the fulcrum point by putting up a narrow shrinkage stope across the panel. If this happened often, this panel method would evolve itself into a series of individual stopes but without the advantage of having been systematized and proper control arranged for in advance.

Advantages of Individual Stope Method

It was decided that a system of mining utilizing individual stopes was best adapted to mining in one lift an orebody whose thickness was 300 ft. or more. Advantages claimed for this method compared with any method in which caving progresses across the orebody are in addition to those indicated above.

1. The area to be caved is confined to a definite area and is surrounded on all four sides by solid ground in the case of the original stopes and by consolidated fill in part or in whole in the case of pillar stopes, compared with an indefinite area, one end of which is recently caved capping and the other end recently undercut ore both in motion, and the sides solid ground in the case of original panels and consolidated fill in the case of the pillar panels. Undoubtedly the support of the individual stope is much better, hence lower maintenance costs, and

less interference of repair work with ore drawing, tending to better tonnage extraction with less dilution.

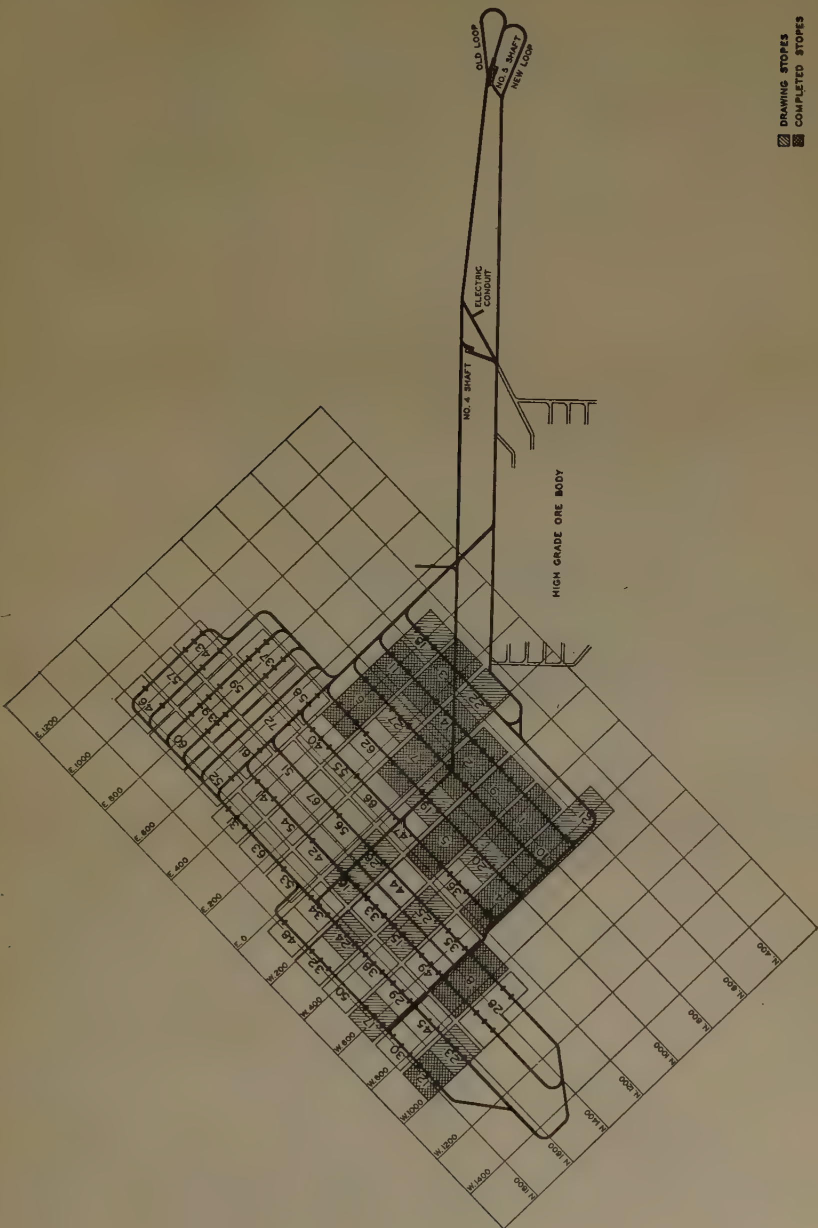

Fig. 2.—Plan of 720-ft. Haulage Level.

2. The order of mining of the stopes is laid out in such a manner that pillar stopes will not be mined until the waste fill along any boundary

has been consolidated for several months. This fill becomes quite a substantial support during this period. The order of the stoping is

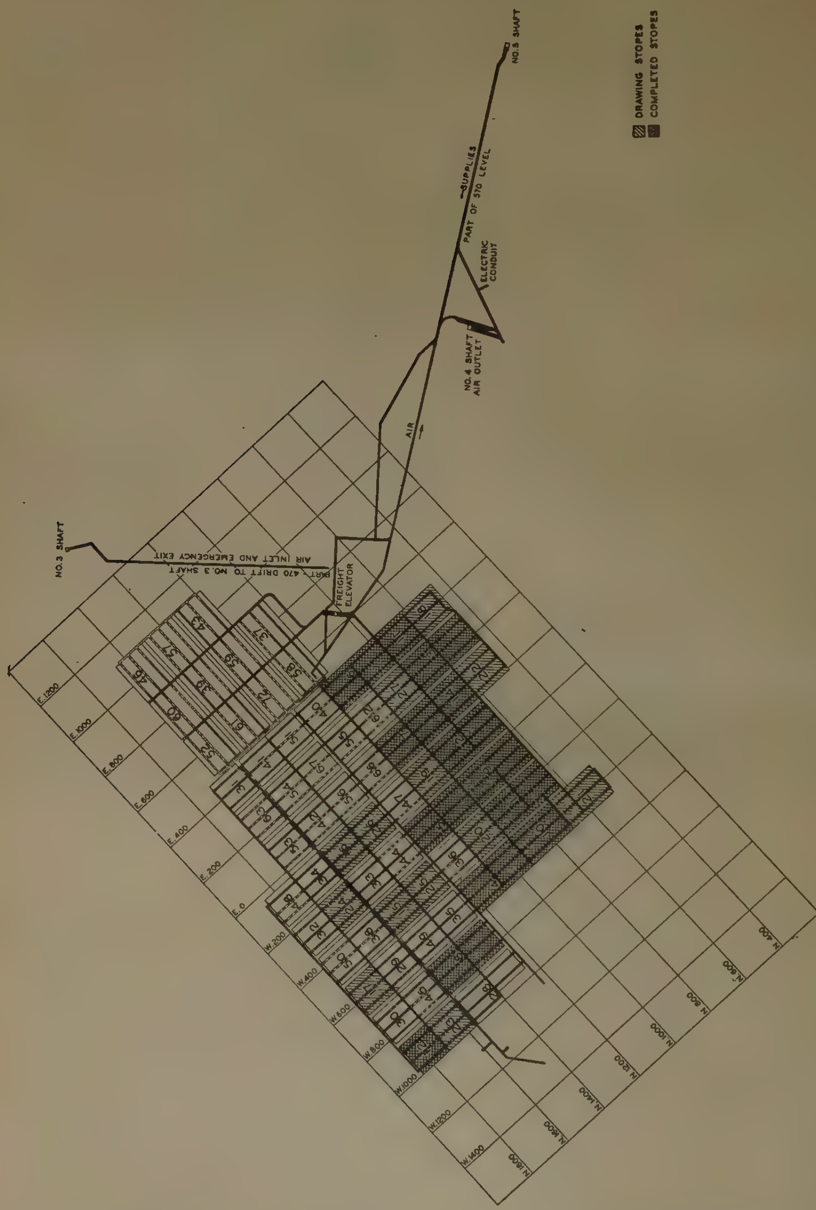

FIG. 3.—PLAN OF 620-FT. AND 635-FT. GRIZZLY LEVELS.

shown in Figs. 2, 3 and 4, which are plans of the 720-ft. haulage level, 620-ft. grizzly level, and 510-ft. boundary caving level. The stopes are mined in the order in which they are numbered on these plans. The

question arises as to whether there would be more dilution of the ore mined from the pillar stopes by waste working in laterally from the adjoining caves of the original stopes than there would be in the panel method. It is thought not, for if the contact between broken ore and waste at the retreating working face of the panel is classified as a waste boundary, the panels would have 21,450 ft. of waste boundary compared with 17,700 ft. of waste boundary for the individual stope method. Therefore, admitting a certain amount of dilution from the waste boundaries, it should be less in the stope than in the panel method. In the stope method, as laid out, there would be mined 25 stopes in solid ground with no waste boundaries, 14 stopes with 25 per cent. waste boundary, 15

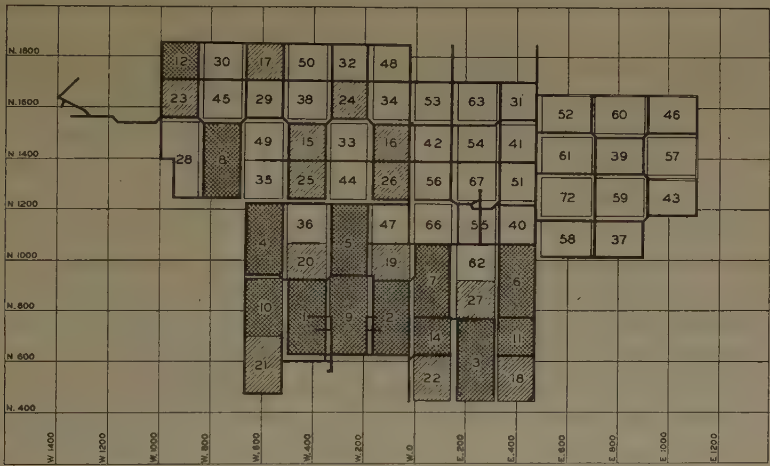

Fig. 4.—Plan of 510-ft. boundary caving level.

stopes with 50 per cent. waste boundary, 14 stopes with 75 per cent. waste boundary and 8 stopes entirely surrounded by waste.

3. In the individual stope method the ore is drawn down evenly over the entire area of the stope, resulting in a horizontal plane of contact between the broken ore and broken waste overlying, and while ore drawing with an inclined plane of contact between ore and waste is satisfactory for low lifts, there can be no question of the superiority of the horizontal plane of contact for high lifts. The inclined-plane method has been used exclusively in the Miami orebody with good results in low lifts since 1913.

4. In the same area of orebody, the stope method provides a greater number of working places and the work is more easily standardized, resulting in a greater tonnage production.

5. In the stope method, the production can be more conveniently distributed over the area to deliver the ore to the various drifts on the haulage level in economical proportions avoiding congestion and delays.

Size of Stopes and Control of Caving

The area of the stope is of the first importance. It should be made large enough so that the ground will cave freely when undercut and small enough so that it will not throw excessive weight on the extraction openings below. In other words, it is a compromise between free caving and low maintenance cost. Based on experience with other caving methods in the Miami orebody, a stope area of 150 by 300 ft. was selected as about right for the average ore. The first nine stopes were mined at this size, and the first eight of these were entirely satisfactory. Stope 9, which was the first pillar stope, gave considerable trouble from excessive weight and the next two pillar stopes were reduced in length, No. 10 to 225 ft. and No. 11 to 150 ft. The development work in stope 12, which was an original stope in particularly weak ground, gave warning of trouble if it were opened out to full size, so it was reduced to an area of 150 by 150 ft.; this size gave such favorable results both in the original and the pillar stopes that it was adopted as standard and the remaining stopes were laid out in this size, with the exception of 11 stopes at the east end of the orebody, which were made 150 by 200 to conform to the closer spacing of the haulage drifts, made necessary because the sills of these 11 stopes were at a lower elevation than the stopes to the west.

The distinguishing feature of this mining method at the time it was adopted five years ago was the high back of ore caved in one lift and the means used for controlling this caving and ore drawing.

The ore is undercut over an area 150 ft. square and allowed to cave by its own weight. If left to itself it might arch to the center and stop caving or it might follow slips or planes of weakness and cave beyond its vertical boundaries. It is essential that the stope should be made to cave to but not beyond its vertical boundaries, and this is particularly necessary when pillar stopes are to be caved and drawn later between the original stopes. With low lifts a satisfactory method of isolating a caving area is by means of a narrow shrinkage stope around the boundary. It was felt that this would be disastrous with a high lift, as the high back of the stope would settle down like a plug or piston and crush the mine workings below and would have the further disadvantage that the broken ore would arrive at the chutes in large blocks unsuitable for drawing. It is desirable to have the back of the stope hang up and become under strain, so that it will become highly fractured and cave down slowly in small pieces and at the same time cave to and not beyond its vertical boundaries. A means of accomplishing this was devised, consisting of driving boundary caving drifts completely around each stope at suitable vertical intervals and putting up raises at all four corners of the stope to avoid lateral arching at these points. These raises also serve a useful purpose as mucking chutes and for ventilation during development. The boundary caving drifts are 7½ ft. high and were originally located

at 30-ft. vertical intervals. This had the effect of weakening the plane of the stope boundary 25 per cent., and when desired in hard ground it can inexpensively be weakened 50 per cent. by drilling and shooting in the back of the drifts. Latterly these drifts are being put in at 45-ft. intervals as a matter of economy and the backs are shot in, thus weakening the boundary plane 33⅓ per cent. Fig. 4 shows one of these boundary caving levels in plan. The following are some of the advantages which are claimed for the method:

1. Delayed caving resulting in finely broken ore and lessened crushing effect on underlying mine workings as described above.

2. The boundary caving drifts serve as exploratory drifts in the early development stages by extending some of these levels beyond the ore limits. In the Miami mine levels 90 ft. apart were thus extended and later on intermediate levels 30 ft. or 45 ft. apart vertically were driven around the stope boundaries. The sampling of these drifts gives valuable information regarding the grade of the ore to be expected as ore drawing progresses.

3. Geological maps of each boundary caving level are prepared, which show the principal slips, inclusions of capping, oxidized ore, etc. These maps are helpful in the control of ore-drawing operations. For example, reference to these maps may show that capping which appeared early at some of the draw chutes was coming from an inclusion low down in the orebody and not from the overlying capping and that ore drawing from these chutes should be continued, whereas without this information they would be sealed. Supplementing these maps, marker blocks are planted in all drifts on all levels at 25-ft. intervals. These blocks are 12-in. wooden cubes and are marked with a copper tag countersunk in the block for protection. This tag contains the elevation of the block and the number of the chute over which it lies vertically. The block is large enough to be held on the grizzly and when it arrives gives accurate information regarding the position from which the accompanying ore has come; also that the stope is caving to its boundary.

4. The boundary caving drifts afford a means of access to the corners of the original stopes, from which the actual fracturing and caving down of the back of the stopes may be observed at successive elevations as it progresses upward, and higher up in the back where the drifts are still open, though usually spalling off due to strain, the side and end boundaries of the stope may be inspected.

Development of Orebody for Mining

The ideal method of development would be first to determine the top, bottom and lateral boundaries of the orebody as well as its assay value by churn drilling from the surface. The shaft would then be

sunk at the most advantageous location from the point of view of safety from ground subsidence, minimum ore haulage underground, and suitability of surface location for mining and treatment plants and surface transportation. With the orebody outlined by churn drilling the complete mining development could be laid out in detail, and the haulage and grizzly levels driven from the shaft at the proper elevations. These two levels should be connected with the shaft, the former for ore haulage and the latter for supplies. All other development raises and levels may be driven from these two principal levels. If the orebody is not previously outlined by drilling, its boundaries should be determined by underground development on the sublevels before the grizzly and haulage

FIG. 5.—STATION ON 720-FT. LEVEL, SHAFT 5, SHOWING CAGE COMPARTMENT OPEN, SKIP COMPARTMENTS SCREENED OFF.

levels are finally located. This is the procedure necessary in the Miami mine, owing to the fact that the drills in the original churn-drilling campaign stopped in material now classified as ore and it was impossible to deepen the holes on account of surface subsidence.

In the development of the Miami orebody the procedure was modified to make use of existing levels connected with the shaft and the 720-of. level, which was the deepest haulage level for the high-grade orebody and already equipped with ore pockets and skip loading mechanism was utilized and extended to serve as the haulage level for the upper lift of the low-grade orebody.

A stoping plan was laid out to fit the orebody so far as developed, so that the exploratory drifting on the upper levels would be done along stope boundaries in order to serve as boundary caving drifts later on. This drifting added considerably to the original area of the orebody and

determined the maximum area to be stoped in the upper lift of the orebody. This is shown in Fig. 2, together with the completed haulage level which was later extended to serve it.

At the present time the preliminary development, which is common to all stopes and consists principally of the haulage level, boundary caving drifts and main entries on the grizzly level, has been completed for the upper lift of the orebody. There remains to be done, just prior to mining each stope, the development required for that particular stope, consisting of transfer raises from the haulage to the grizzly level, the grizzly drifts and raises and chute sets. The preliminary development of the lower lift of the orebody is in progress and, illustrating the use of the boundary caving drifts for exploratory purposes, these have been advanced to date a distance of 27,640 ft., while the haulage level has only been advanced 1616 ft. and the grizzly level has not yet been started.

Development work varies largely from year to year, depending upon whether a new lift of ore is being prepared and a large proportion of preliminary development being done. To equalize this a fixed charge per ton for development is made currently and any excess over this is charged to suspense development account and any deficiency is credited to this account. This final charge of 10 c. per ton is estimated to cover the development of the entire orebody (see Table 3). The development for the year 1928 amounted to 52,543 ft. of drifts and 32,076 ft. of raises, and additional footage of 43,032 ft. of drifts and raises charged to stoping account, making a total footage for the year of 127,651.

Method of Mining and Preparation for Ore Drawing

The method of mining is best illustrated by Fig. 6, which is an isometric drawing of an individual stope 150 by 300 ft. The same illustration applies to the half-size stope now in use.

The spacing of the various drifts and raises in the development of the orebody is determined primarily by the spacing of the draw points under the broken ore. The Miami ore breaks up finely and tends to pack and when drawn tends to pipe up vertically with little spread beyond the draw points. This condition makes it necessary to space the draw points as close together as economy and the necessity for maintaining supporting pillars between will permit. With increasingly coarser ore the spacing and size of chute openings should be increasingly greater. It was decided to space the draw points 12½ ft. apart in both directions. This spacing determined a spacing of 50 ft. apart for the grizzly drifts and 25 ft. apart for the grizzlies or tops of transfer raise branches. The spacing of the haulage level drifts is partly determined by this and partly by the distance between this level and the grizzly level and the angle of the inclined transfer raises. In the illustration the haulage level

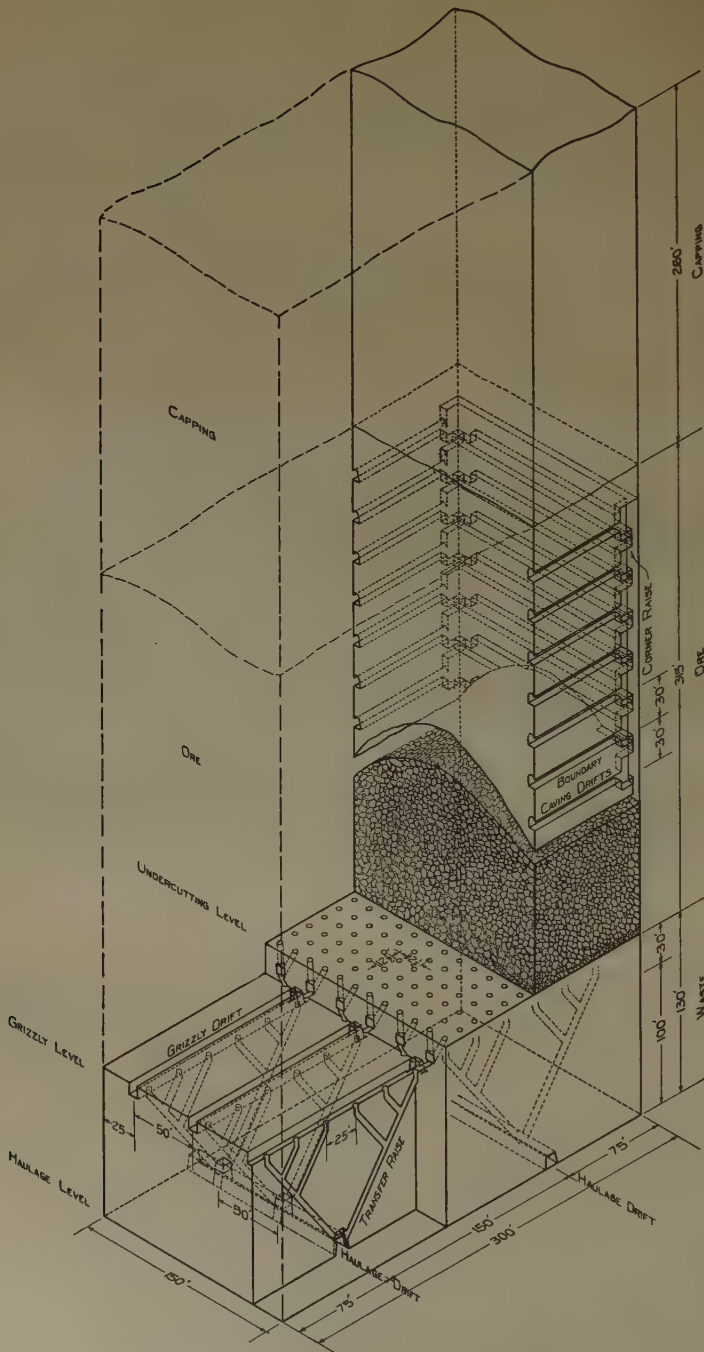

Fig. 6.—Isometric drawing of stope 150 by 300 feet.

is 100 ft. below the grizzly level, the drifts are spaced 150 ft. apart and the transfer raises have three branches 25 ft. apart.

A different arrangement is used for the 11 stopes at the east end of the orebody where it dips lower and where the grizzly level was lowered to a level 65 ft. above the haulage level. There the haulage drifts are spaced 100 ft. apart and the transfer raises have two branches 25 ft. apart. A comparison of the two arrangements would be about as follows: In bad ground requiring timbered raises the costs of haulage drifts and raises would be about the same at $0.96 per square foot of stope area served, but in good ground with untimbered raises the 150-ft. spacing would be cheaper at $0.55 compared with $0.65 per square foot for the 100-ft. spacing. Other advantages and disadvantages of the 150-ft. spacing of haulage drifts with longer raises compared with 100-ft. spacing and shorter raises would be:

Advantages of 150-ft. Spacing

1. Greater capacity for ore storage in raises.
2. Less danger in blasting at either end of raise to workmen at other end.
3. Faster to develop owing to less drifting and chute construction, which is slower than raising.

Disadvantages of 150-ft. Spacing

1. Fewer haulage drifts and more train congestion.
2. Fewer raises with decreased check or control of ore drawing.
3. Greater flow of ore through each raise, resulting in greater damage to lining.
4. Longer raises increasing hazard of driving.
5. Increased blasting of hung-up raises.
6. More difficult to inspect long raises.

Sequence of Operations

The sequence of operations in preparing a stope 150 by 150 ft. for mining after the preliminary development as outlined above has been completed is as follows: Three pony sets are installed over the haulage drift, the middle one directly under the center of the stope and the others 50 ft. on either side. Six transfer raises, inclined at 55° 20′ from the horizontal, are driven at right angles to the haulage drift from both sides of the three pony sets. At the same time three grizzly drifts (Fig. 7) are driven at right angles to the haulage level drifts on the level 100 ft. above it and vertically over the transfer raises. These three drifts are driven at both ends from the fringe drifts, which are part of the grizzly

level preliminary development. When connected with the three branches of each of the six transfer raises, these three drifts with fringe drifts connecting their ends constitute a stoping unit of the grizzly level. The

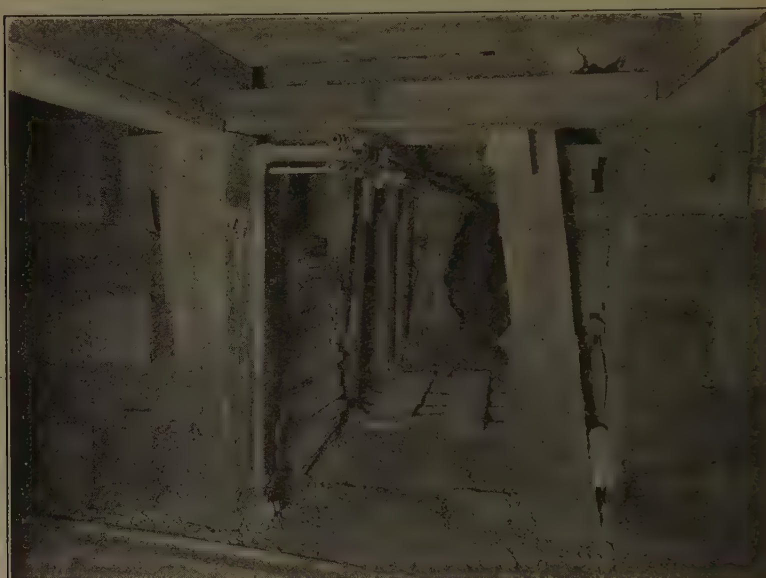

FIG. 7.—LOOKING DOWN GRIZZLY DRIFT FROM FRINGE DRIFT IN STOPE 19.

620-ft. grizzly level, shown in Fig. 3, is a most important level; 50 per cent. of the entire underground force is employed on this and the undercutting level served by it, and 85 per cent. of all mine supplies are used

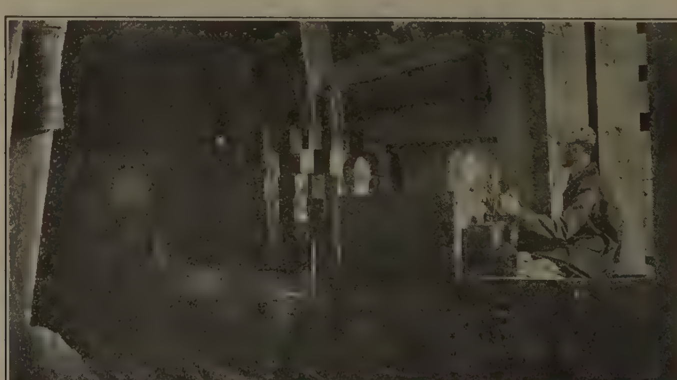

FIG. 8.—SUPPLY TRAIN ON 620-FT. GRIZZLY LEVEL.

on or distributed from this level. During actual mining it is the base of operations for undercutting and ore drawing. This concentration of operations on the grizzly level necessitated its careful planning, partic-

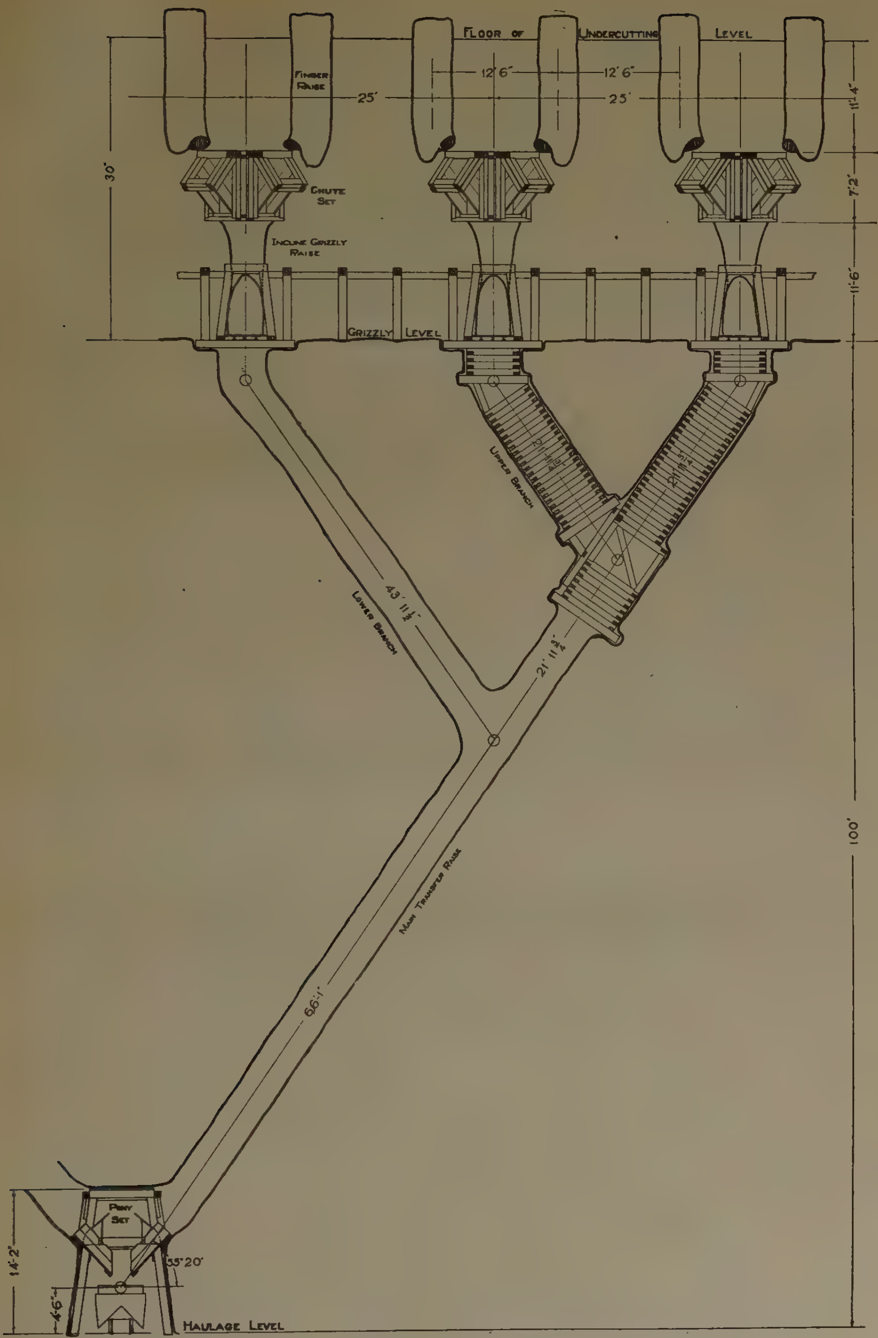

FIG. 9.—VERTICAL PROJECTION OF ORE-TRANSFER RAISE SYSTEM.

ularly with a view to providing adequate ventilation and efficient distribution of supplies. Owing to the necessity for more or less continuous chute blasting on this level, it is aimed to circulate 2000 cu. ft. of air per minute through each active drift. This circulation is regulated by ventilating doors at either end of the grizzly drift.

Following the connection of the tops of the transfer raises with the grizzly drifts, grizzlies are installed over these openings. The grizzlies consist of 45-lb. rails spaced 12 in. apart across the drift, supported on 10 by 10-in. stringers. An important detail in this connection is to provide substantial support for the grizzly and drift floor at this point by topping the inclined raise with a short vertical section, and it is advisable to insure a clean-cut connection by sinking a winze from the floor of the drift one round and leaving a long drill projecting down from the center of the bottom as a guide for the raise men in making the connection.

The next step is to drive grizzly raises from both sides of each of the 18 grizzlies. These are driven at right angles to the grizzly drift; they are inclined at 42° for a distance of 14 ft. and thence vertically 10 ft. The inclined section of these raises is made small (3½ by 3½ ft.) to resist crushing, and the upper part of the vertical section is enlarged to accommodate the chute set. The correct orientation of these chute sets is important, as upon it depends the correct spacing of the draw points above. The sills for all of these sets are placed by a special crew and checked by the development engineer. As the weight developed at this point is considerable, these chute sets are substantially built and reenforced on all four sides with additional caps and batter posts, which also form a part of the chute construction and help to maintain the spread of the draw points. Four finger raises 3½ ft. dia. are driven from these chute openings on line with the chutes; inclined to a point 8.85 ft. horizontally from the center of the set and thence vertically to a horizontal plane 30 ft. above the floor of the grizzly level, which they intersect at points 12½ ft. apart east and west and north and south. Contrary to previous practice, the chute sets are oriented at 45° with the grizzly raise instead of square with it. This brings the chute openings in the set on line with the draw points and avoids the necessity of turning the finger raises to reach these points. This is an important detail, which makes for accuracy and simplicity in spacing and maintaining the draw points. The details of this development from the haulage level to the finger raises are shown in Figs. 9 and 10.

The stope development work is started at the end of the stope where caving is to begin and progresses toward the other end. As soon as a sufficient number of finger raises have been put up to the plane 30 ft. above the grizzly level, the undercutting level is started from the tops of these raises, utilizing one or several of them as supply raises and manways and those nearest to the work for muck. The undercutting level

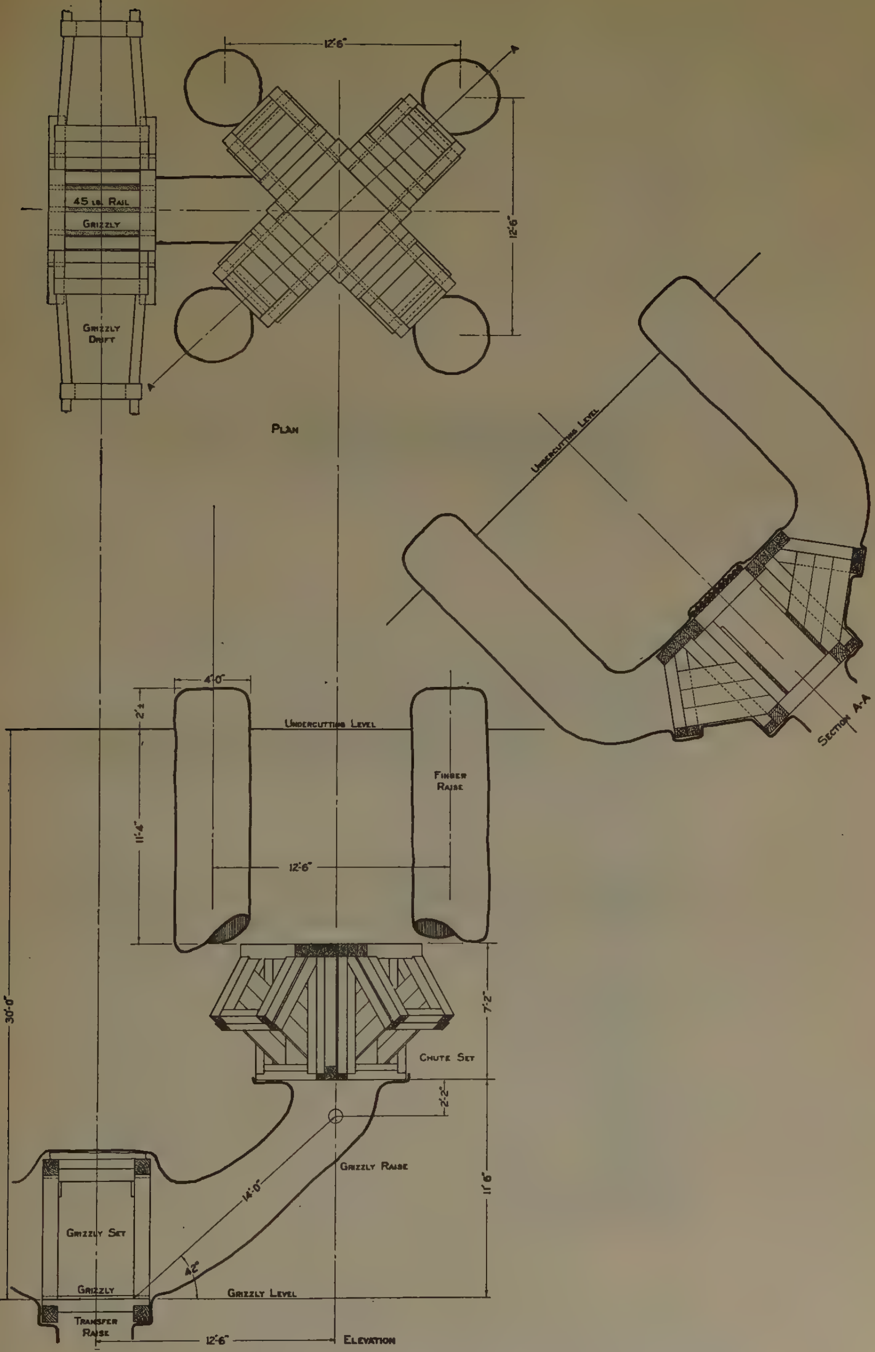

Fig. 10.—Location and orientation of chute set.

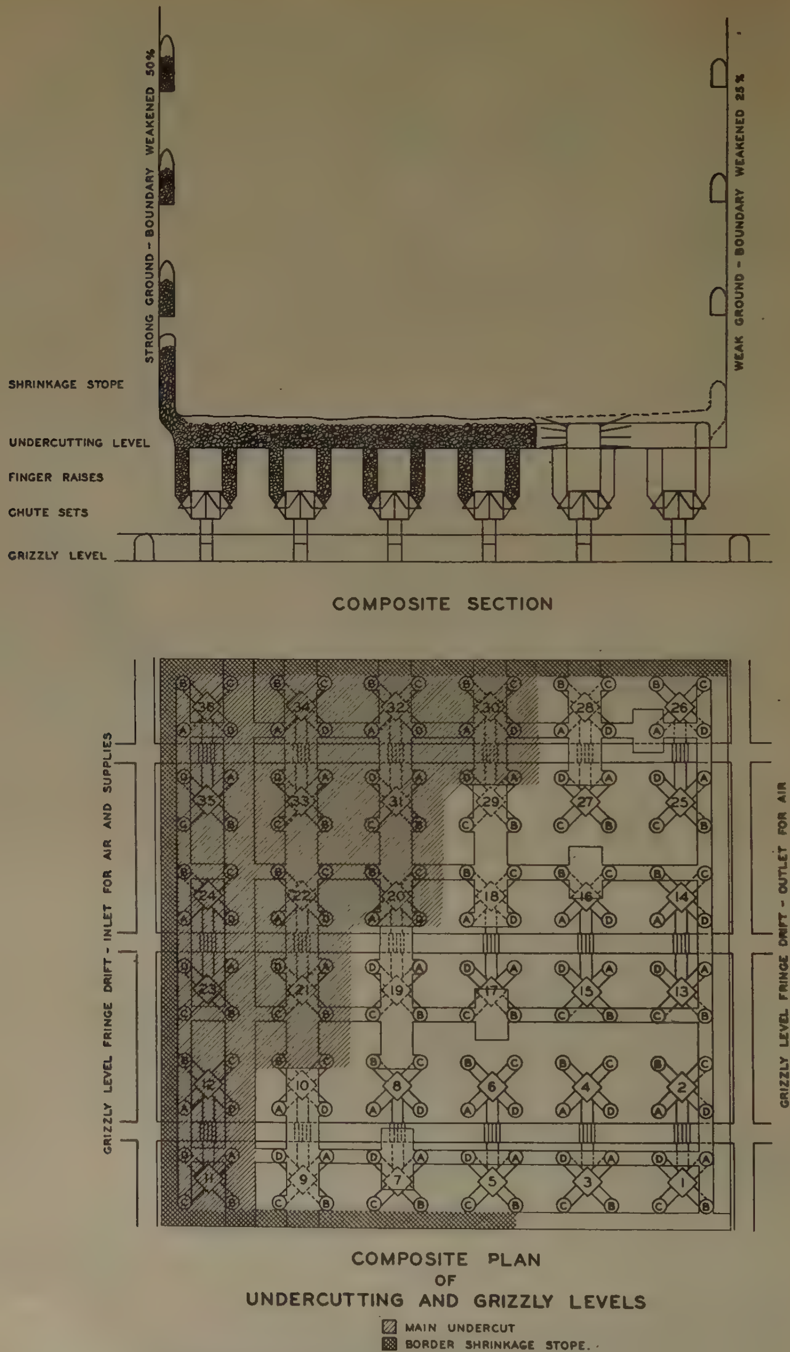

FIG. 11.—METHOD OF UNDERCUTTING.

and its relation to the grizzly level, chute sets, and draw points is shown in plan and section in Fig. 11. The chute sets are numbered from 1 to 36 and the draw points are lettered. The undercutting level is opened by driving four drifts of small cross-section parallel to the grizzly drifts through every third line of draw points 37½ ft. apart and equidistant each side of the central grizzly drift. These drifts are connected at both ends by fringe drifts. Undercutting is started by putting up a shrinkage stope two or three rounds from one of the fringe drifts and is usually carried along the side boundaries and other end of the stope as undercutting of the main body progresses. The main body is undercut from large drifts 8 ft. wide, which are driven at right angles to the small opening-up drifts. This work is carried back diagonally, as shown cross-hatched on the plan, and ends at the first chute set. The large drifts are not driven until needed for blasting, in order to avoid premature crushing. On the plan, 9, 10, 19, 18 and 28 represent drifts which are ready to be drilled up and blasted in, while 7, 17, 16 and 26*A* are just being driven. The sides and backs of the drifts are drilled and blasted, which complete the undercut. The tops of all finger raises are funneled as undercutting progresses. The undercutting of a stope, once it is started, is completed as rapidly as possible, for obvious reasons. The small opening-up drifts, however, owing to their small cross-section and thick pillars between, may be driven in advance and held without damage until necessary to start undercutting operations, thus promptly providing numerous working faces for the undercutting crews.

Usually ore drawing is not started until the stope is completely undercut, although there are exceptions to this rule in some cases, when drawing is started to avoid excessive packing of soft ore and to ease weight on the chute sets and grizzly level below.

The work preparatory to ore drawing described above is done according to written schedule prepared a year in advance for each stope.

A typical stope schedule and progress record appears in Table 1. The schedule is shown in the first and last columns and the progress is recorded in the intermediate columns.

This schedule serves as a guide to the engineer in charge in assigning the proper number of workmen to each job. He also sees that the men best qualified for the different classes of work are assigned to that work. The practice throughout the mine is to build up specialists for each job. It enables the men to make more money at jobs at which they are most skillful and expedites the work at less cost to the company and makes advance estimates of progress more reliable.

The amount of development and preliminary stoping that is necessary each month in order to maintain the normal monthly production of 525,000 tons from the part of the orebody served by the 720-ft. haulage level is shown in Table 2.

TABLE 1.—*Developing and Undercutting Schedule and Progress Record of a Typical Stope*

	Scheduled to Start	Started	25 Per Cent. Complete	50 Per Cent. Complete	75 Per Cent. Complete	Complete	Scheduled to Complete
Transfer raises	1-1-29	12- 2-28	12-29-28	1-28-29	2-28-29	5-15-29	6- 1-29
Grizzly-level drifts	4-1-29	3-20-29	4- 6-29	4-23-29	5- 7-29	5-25-29	6- 1-29
Grizzly raises	5-1-29	5- 5-29	5-15-29	5-21-29	5-29-29	6- 9-29	6-15-29
Chute sets	6-1-29	6-10-29	6-25-29	7- 8-29	7-25-29	8- 5-29	8- 1-29
Finger raises	7-1-29	7- 5-29	7-21-29	7-29-29	8- 8-29	8-20-29	9- 1-29
Undercutting-level drifts	8-1-29	8- 3-29	8-23-29	9- 8-29	9-22-29	10- 4-29	10- 1-29
Undercutting-level mining	10-1-29	10-14-29	10-25-29	11- 4-29	11-12-29	11-18-29	11-15-29

TABLE 2.—*Development and Preliminary Stoping per Month*

CLASSIFICATION	UNITS
Haulage-level drifts, feet	345.8
Haulage-level chute sets	3.5
Transfer raises, feet	1,105.3
Grizzly-level drifts, feet	688.1
Grizzly-level raises and chute sets	37.7
Boundary caving drifts, feet	2,292.6
Boundary caving corner raises, feet	299.3
Finger raises	150.6
Undercutting-level drifts, feet	897.4
Undercutting-level drilling and blasting in, square feet	23,528.9
Boundary caving drifts drilling and blasting in, square feet	2,292.6
Boundary caving raises drilling and blasting in, raises	7.6

In Table 3 are shown the tons served by each unit of the various classes of development and preliminary stoping, together with costs per unit and the cost per ton for this work over the period 1925–28, inclusive. It would appear from this that the uniform charge for development of 10 c. per ton of ore drawn is ample.

ORE DRAWING

There are two principal objectives in this operation: (1) To draw a maximum of the ore tonnage with a minimum of dilution by waste capping. To accomplish this an effort is made to draw the ore down evenly so that the contact between the broken ore and broken capping will be an even plane and preferably a horizontal one. (2) To regulate the drawing to avoid, if possible, or to relieve damaging weight on the extraction openings below the broken ore; with the object of reducing maintenance and ore-drawing costs and interference with the predetermined order of ore drawing.

TABLE 3.—*Development and Preliminary Stoping Requirements and Costs for Orebody Served by 720-ft. Haulage Level**

39,968,411 Tons

	Average Estimated Tons Served Per Unit	Cost per Unit 1925–28 Inclusive	Cost per Average Estimated Ton in Place, Based on 1925–28 Costs
Haulage level, per foot	1,518	$19.950	$0.01315
Haulage-level chute sets, per set	148,031	325.606	0.00220
Transfer raises, per foot	475	8.496	0.01789
Grizzly-level drifts, per foot	763	10.700	0.01402
Grizzly-level raises and chute sets, per set	13,917	336.701	0.02419
Boundary caving drifts, per foot	229	7.033	0.03071
Boundary caving corner raises, per foot	1,754	3.528	0.00201
Total development			$0.10417
Finger raises, per raise	3,486	32.789	0.00941
Undercutting-level drifts, per foot	585	2.996	0.00512
Undercutting-level mining, per sq. ft	22.3	0.308	0.01378
Drilling and blasting boundary caving drifts, per foot	229	1.119	0.00489
Drilling and blasting boundary, caving corner raises, per raise	68,439	87.407	0.00128
Total stoping			$0.03448

* Based on tonnage extraction of 12,710,378 from stopes completed to date, equivalent to 115.15 per cent. of tonnage estimated, the cost per ton of ore extracted for development and stoping would be reduced to $0.0905 and $0.0299 respectively.

Ore drawing as it actually works out in practice is a compromise between the requirements of these two principal objectives. Immediately after undercutting of the stope is completed ore drawing is started. It is usually slow work to begin with, because the first ground caved usually arrives at the chutes in fairly large pieces. Sometimes the stope back hangs up and drops only a very little ore for several weeks, but this is unusual. As drawing progresses the ore becomes more finely broken, partly due to the strain to which it is exposed in the back of the stope and partly due to the crushing action in the broken mass on its way down to the chutes.

To maintain the normal daily tonnage of approximately 18,000 tons it is necessary to have 13 or 14 stopes on the drawing schedule. Of this number, the stope or stopes about to be finished and those just beginning produce a comparatively small tonnage. In addition to the ore produced from actual ore-drawing operations there is a small tonnage produced from stopes in preparation for ore drawing and from mine development.

Table 4 shows a typical distribution of the ore produced for one months' actual operations in which the mine was worked $28\frac{2}{3}$ days.

TABLE 4.—*Typical Distribution of Ore Produced in One Month*

$28\frac{2}{3}$ Days of Operation

	Average Tons per Day		Average Tons per Day
Ore drawing stopes		Forwarded	9,457
No. 15	1,454	No. 23	1,787
16	1,355	24	1,199
17	1,296	25	1,524
19	1,390	26	1,283
20	776	27	1,380
21	2,061	28	555
22	1,125		
Forward	9,457	Total	17,185
Six stopes in preparation			199
Development			396
Total tons per day			17,780

In that tabulation, 17,185 tons were produced from ore-drawing operations in 13 stopes, which was equivalent to an average of 1322 tons per stope, or 17.06 sq. ft. of area required per ton of ore produced daily. The corresponding average thickness of solid ore drawn daily would be 0.733 foot.

The ore drawing is under the supervision of three stope engineers on day shift, who inspect the stopes daily and issue ore-drawing orders to four draw bosses and five chute sealers on each of the three shifts. From these orders the draw bosses issue orders to the different crews under their direction, listing the chutes to be drawn in groups of chutes which empty into the same transfer raise. The chute sets are numbered from 1 to 36 and the four chutes in each set are lettered A, B, C and D. The A chutes are drawn by one shift, B chutes by the next shift and so on in rotation. The drawing is done by crews of two men in each, consisting of a chute blaster who receives miner's wages and a chute tapper who receives mucker's wages. The chute blaster draws the ore from the chute in the chute set and the tapper keeps the grizzly clear on the grizzly level. The blaster does all ordinary chute blasting required, but in addition to this there is a special crew of chute blasters and chute drillers who bring down specially difficult chutes or hung-up raises. All chute blasting except the blasting of loaded drill holes is done by means of small hand batteries. The advantages of electric caps over fuse and ordinary caps are:

1. Possibility that a bomb already spit may fall down the transfer raise to the haulage level is avoided.

2. Bombs can be placed in high chutes by means of blasting sticks to better advantage.

3. No interference is caused by smoke from fuse.
4. Less delay is caused in waiting for missed holes.
5. Man who places charge is not endangered by possibility of its explosion while he is too near.

The tapper crew will usually draw from 12 to 15 finger raises per shift, blasting from five to eight times and drawing approximately 400 tons per shift. Normally 50 tons is drawn daily from each chute listed on the draw orders. The aim is to draw this amount in rotation from each chute in the stope, but variation in this regular routine is brought about by various causes, such as the appearance of capping in a chute; necessity for repairs; appearance of weight on the timbers, requiring the chutes to be drawn to relieve this weight whether listed to be drawn or not; variation in drawing to meet requirements as to grade of ore; changes in order of drawing to maintain proper distribution to haulage level for economic operation of the trains. Each stope engineer inspects each day all chutes listed on the draw sheet and receives reports from draw bosses and sealers as results of their inspections. He notes the condition of the ore in all the chutes; removes from the draw list chutes in which capping shows; lists chutes that need drilling or blasting and chutes or chute sets or grizzly level timbering that require repair, and orders this done in the order needed.

Any chute in which capping shows up prematurely is sealed for several weeks before being placed on the drawing list again. In the meantime the chutes surrounding it may be sealed and later drawn, which tends to break up the wall of the pipe, after which the chute with capping may usually be drawn again with satisfactory results.

The stope engineer carries in his notebook graphs representing the tonnages drawn from the chutes along each row. These are posted every two days so that as he inspects the stopes he has reference to an accurate record of the stopes' condition. Office records of each stope are kept. These consist of similar graphs of the tonnages drawn along each row of finger raises; there are twelve graphs for each stope, which are posted up to date semimonthly.

Drawing is continued from each chute until the grade of the ore drops to a point that is not sufficiently profitable. This usually does not occur until after 100 per cent. of the expected ore has been drawn. This cut-off point is fixed by the minimum profit required, which may vary, depending on the cost of operation in the particular stope, market price of the metal, capacity of treatment plant, and the total profit requirements. It is also affected by whether there is another lift of ore to be drawn from beneath it or not. In case there is, the grade of the ore at which drawing stops may be higher, as there will be another opportunity to recover it from below.

Although the large capacity of the transfer raises makes ore drawing fairly independent of transportation on the haulage level below, the method of measurement of ore drawn requires close cooperation between these two operations, and this is maintained by telephone service between the draw bosses and the train dispatcher on the haulage level, who issues orders to all trains on their return trips from the shaft.

Drawing of the chutes by the tapper crews according to orders issued by the stope engineers is controlled by keeping all chutes sealed and only permitting the tapper crews to break seal and draw chutes which have been listed by the draw bosses for drawing. It is the duty of the draw bosses and chute sealers to keep all chutes sealed and to seal those listed immediately after they have been drawn. If any have been drawn toward the end of the shift, which they have been unable to seal, they leave a list of these for the draw bosses and sealers on the next shift. This sealing is done by tacking a cardboard strip initialed by the sealer to the side and bottom door of the chute so that the door cannot be opened without breaking the seal.

In order to draw the ore down evenly it is necessary to have a method of measuring the ore drawn and of recording this so that there is a continuous record to indicate the chutes that should be drawn next. When this method of mining was put in operation, two systems of measuring the ore drawn from each chute were tried out, one of them being the one now in use. The other system consisted in converting the tops of the transfer raises and branches into measuring pockets by installing doors in them about 20 ft. down from the top. These doors were opened or closed by means of a cable from the grizzly level. This system was abandoned for two principal reasons:

1. The ore tended to arch in the measuring pocket after the door was opened, which caused trouble and delay.

2. The ore drawing was slowed down considerably, owing to the necessity for stopping and starting the flow of ore several times as the measuring hopper became nearly filled, and demonstrated that it is the wrong principle in ore drawing to attempt to draw an exact amount. It is better to draw rapidly approximately the amount required and measure it exactly afterwards.

The method of ore measurement and control now in use utilizes the ore cars on the haulage level as units of measurement, but as 24 finger raises discharge into each transfer raise, and as it is essential to record the tonnage that comes from each individual finger raise, it is obvious that in order to utilize the ore cars as measuring units the finger-raise drawing and transfer-raise drawing must be coordinated. This is brought about by listing the chutes to be drawn by the chute tappers in such a way that they draw one chute only which empties into one transfer raise and so on with succeeding transfer raises, and another chute empty-

ing into the first transfer raise is not drawn until this raise has been drawn on the haulage level. The chute tappers record on their lists the exact time of starting and stopping the drawing of each finger raise and the motorman on the haulage level records the time and order of drawing the various transfer raises on the haulage level. At the end of the shift the tapper's records are recorded on the draw boss' report, which shows by a heavy line the time of starting and stopping the drawing of each chute; above this line is recorded the tapper's estimate of the tons drawn, and below it the number and letter of the chute. From the haulage motorman's report the tonnage clerk at the mine office enters on the

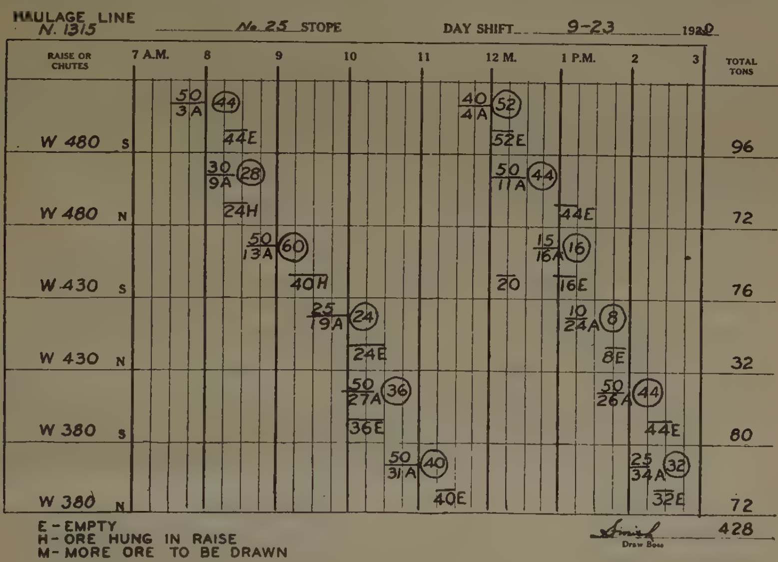

FIG. 12.—REPORT OF DRAW BOSS.

draw boss' report, in the lower half of the space on which the draw boss' record is posted, the time and tonnage drawn from the transfer raise on the haulage level, indicating by letter whether the raise was drawn empty, whether muck was left in it or whether it was hung up. By coordinating the time of drawing the finger raises with that of drawing the transfer raises, an exact measurement of the ore drawn from each finger is obtained for about 75 per cent. of the ore drawn. The remaining 25 per cent. has to be split between two finger raises, owing to the overlapping of drawing or hanging up of the transfer raise, or other causes. This 25 per cent. is apportioned by the tonnage clerk to the draw chutes contributing to it. His final tonnage for each chute appears in the circle to the right of the tapper's estimated figure on the draw boss' report. After the haulage-level tonnage is posted on the draw boss' report, this is returned to the draw boss so that he may see how close the chute tappers are

estimating the tonnage drawn. The tappers have become expert in this estimation, based on the appearance of the flow of ore as it passes out of the chute and inspection of the raise into which it empties. It is believed that this method of measuring the ore drawn from the individual finger raises is reasonably accurate and is entirely satisfactory for the purpose of control of ore drawing. A sample of the draw boss' report appears as Fig. 12.

Up to date 13 stopes have been completely drawn. Nine of these were original stopes surrounded on all four sides by solid ground and four were pillar stopes, two of which were adjoined by broken waste on two sides and one end, and two adjoined waste on one side and one end. These 13 stopes were estimated to contain 11,038,070 tons of ore assaying 1.026 per cent. copper. They produced 12,710,378 tons, assaying 0.912 per cent. copper. Table 5 gives a tabulation showing the tonnage, grade and copper extraction from these 13 stopes, the same figures for the best and for the poorest original stope, and for the best and for the poorest pillar stope.

Table 5.—*Tonnage and Grade Extraction*

Partitions Not Included in Expectancy

	Expectancy		Mined	
	Tons	Copper, Per Cent.	Tons	Copper, Per Cent.
Total of 13 completed stopes	11,038,070	1.0260	12,710,378	0.9124
Best original stope (2)	998,016	1.0388	1,210,424	1.0091
Best pillar stope (11)	319,560	1.0640	387,827	0.9348
Poorest original stope (7)	1,071,535	0.8701	1,053,153	0.7786
Poorest pillar stope (9)	1,098,313	1.1067	1,025,032	0.8995

	Percentage Extraction		
	Tonnage, Per Cent.	Grade, Per Cent.	Copper, Per Cent.
Total of 13 completed stopes	115.15	88.93	102.40
Best original stope (2)	121.28	97.14	117.81
Best pillar stope (11)	121.36	87.86	106.63
Poorest original stope (7)	98.28	89.48	87.94
Poorest pillar stope (9)	93.33	81.28	75.86

The extraction of more than 100 per cent. of the estimated copper content of the ore may be due to drawing some of the partition ore, and to copper not included in the estimate, coming from overlying capping or gob.

The best pillar stope, 11, was bordered by waste on one end and one side, and the extraction from it compares favorably with the best original stope, the tonnage extraction being almost identical, while the grade extraction is 10 per cent. lower. The poorest pillar stope, No. 9, is the poorest of the lot, but it was the first pillar stope mined. It was 150 by 300 ft. in plan, and so much weight developed during its extraction that results were far from good. The experience gained here resulted in reducing the size of the stopes to one-half their original size. No. 11, the best pillar stope, was a one-half size stope.

This explanation is made in view of the fact that the question has been raised by many engineers as to the results that might be expected in mining these pillar stopes. Experience gained in previous caving methods using the panel system, retreating across the entire orebody, indicated that it was advisable to leave a thin partition of ore between the original stopes and the pillar stopes, or, in other words, to leave a greater distance between the border draw point of the pillar stope and the border draw point of the original stope adjoining. Acting on this, a partition of 15 ft. was left between the sides of all stopes and no partition was left between the ends of the two southern rows of stopes, while end partitions were left between the three rows of northern stopes. The original partitions are 15 ft. thick, and some of the later ones 7½ ft.; the tonnage in the partitions amounts to 11.0 per cent. of the total tonnage served by the 720-ft. haulage level. This tonnage is excluded in figuring development and mining costs, but drawing records up to date indicate that a part, at least, of this tonnage is being recovered.

In drawing the pillar stopes, there is no serious trouble experienced from waste drawing in from the adjoining previously mined original stopes. It has been the experience in the past that the drawing spreads very little and tends to pipe up vertically once the ore has been broken. This characteristic necessitates the close spacing of draw points, but at the same time protects the operation from waste rock drawing in laterally. The waste capping which has settled down into the original stopes becomes well packed and consolidated, due to the pressure exerted by a vertical column of 300 or 400 ft. of this material, and this waste filling constitutes substantial side support in mining the pillar stopes. Before the first pillar stope, No. 9, was mined, crosscuts were driven into the adjoining fill of the original stopes 1 and 2 on the 510-ft. level, which is 80 ft. above the sills of these stopes, and this material at this point appeared to be as solid as the original ore along the sides of pillar stope 9, through which approach drifts were driven.

The following statistics represent the average results of drawing the 13 complete stopes, producing 12,710,378 tons of ore; 79.59 per cent. was drawn clean, that is, with no dilution from capping. These 13 stopes contained 802 chute sets, and of these 314, or 39.2 per cent.,

during the operation of drawing required to be completely replaced, some of them several times, the total replacements amounting to 862, or 107.5 per cent. of the original installation; and in addition to this, there were 969 minor repair jobs in these 802 original sets. In 341 sets, or 42.5 per cent. of the total original installation, no repairs whatever were required. Of the total number of shifts worked on maintenance of these chute sets, 79.1 per cent. were for complete repairs and 20.9 per cent. were for minor repairs.

Underground Transportation

Ore is hauled from the various transfer raises outletting to the haulage level in trains consisting of 35 cars of 86 cu. ft. capacity, carrying 3.6 tons of ore each. The train crew consists of one motorman and two brakemen. These trains are pulled by two 6-ton trolley locomotives equipped

Fig. 13.—Ore train drawn by two 6-ton trolley locomotives operating in tandem.

with two 250-volt ball-bearing motors each (Fig. 13). The rated drawbar pull per motor is 3000 lb. and the rated speed is 6.5 miles per hour. All motors are equipped for tandem operation at either end, or for use as single motors. In switching and supply service, they are used as single motors, while in the ore service they are operated in tandem, both locomotives being controlled from the lead locomotive.

The track is 24-in. gage and 45-lb. rail, with the exception of the approach of 1200 ft. to the shaft, where 70-lb. rails are in use on account of the heavy traffic over this stretch. The grade is 0.4 per cent. in favor of the load. Curves have a radius of 41 ft. 3 in., and all switches are ground-throw spring switches, with manganese cast frogs. The cars

are gable-bottom side-dump. The doors are locked and unlocked by means of one lever connected to both doors through a series of links. In unloading at the ore pocket (Fig. 14), the levers are hand tripped by one brakeman and locked by the other, while the train is moving over the pocket at a speed of about 2 miles per hour. The car frame is cushioned on the axles; the wheels are Timken equipped. All car repairs are done on the surface, although a thorough car inspection is made underground twice a month and bearings are greased once in 60 days.

The production of 18,000 tons per day is handled by six trains operating on ore each of the three shifts, with one additional service train on

FIG. 14.—UNLOADING ORE CAR AT SHAFT POCKET, 720-FT. HAULAGE LEVEL.

each shift to gather and haul development muck. The average haul from the transfer chutes to the shaft pocket is 0.96 mile one way. It requires an average of 30 min. to load the trains; they are hauled to the station, dumped and returned to the loading chutes in 26 min., or an average of 56 min. per round trip. Each train hauls between 950 and 1000 tons per shift. All important tracks are one way only, and trains are not backed except when loading from the chutes. The trains are loaded at the loading chutes through steel arc-type gates (Fig. 15) operated by one of the brakemen in the pony set above the haulage drift. The other helper is in the drift below; he takes a sample of the ore as the car is loaded, and signals the motorman to move the train when ready.

Train movements are controlled by means of approximately 40 hand-operated signal lights controlling 10 individual blocks. Contact

is made with each motorman each trip through a train dispatcher whose station is on the return line from the shaft. This dispatcher issues written orders originating from either the train bosses or the draw bosses, with whom he is in telephonic communication.

The ore pocket at the shaft has a capacity of 800 tons.

Fig. 15.—Rear end of ore train passing through loading drift on 720-ft. haulage level.

The cost of ore haulage, including loading, hauling, dumping, maintenance of track, equipment and haulage drifts, is about 5 c. per ton-mile.

Ore Hoisting

The ore is hoisted to the crusher plant bins a vertical distance of 811 ft. in 10½-ton skips, which are loaded from the ore pocket through an automatic measuring cartridge operated by the skiptender, whose station gives him a view of the skips being loaded. He also puts the skips in motion when loaded, through a push-button control, after which the acceleration, retarding, and dumping of the skip are automatic, no hoist engineers being required. The maximum speed of the hoist is 2250 ft. per min. and the trip is made in 38.5 sec. The time required for loading and dumping is 3 sec. The maximum capacity of the hoist is 950 tons per hour, and the power required is approximately 1 kw-hr. per ton hoisted.

The ore hoist as originally installed had a maximum capacity of 12,000 tons per day. When it was decided to increase production to

18,000 tons per day, the hoist was brought up to this capacity by increasing the speed 50 per cent. from 1500 ft. to 2250 ft. per minute by doubling the power input. The hoist being motor-driven through a pinion and gear, this was brought about by increasing the size of the pinion on the original 1400-hp. motor and installing a similar motor and pinion on the opposite side of the gear wheel; at the same time duplicating the original motor-generator flywheel set. At the same time that this was done,

TABLE 6.—*Test Data No. 5 Skip Hoist*

Time of Test 10:09 a.m. to 11:10 a.m. Aug. 20, 1929

Drum speed r.p.m., east skip	75.3
west skip	75.3
Hoist motor speed, east skip	331
west skip	331
Acceleration time, seconds, east skip	11.4
west skip	11.3
Retardation time, seconds, east skip	12.5
west skip	13.0
Running time, seconds, east skip	35.0
west skip	35.2
Rest period, seconds, east skip	3.0
west skip	3.0
Total time of trip, seconds, east skip	38.6
west skip	38.6
No. 1 motor-generator set speed, no load, r.p.m.	747
No. 2 motor-generator set speed, no load, r.p.m.	747
Operating speed No. 1 motor-generator set, r.p.m. max	722
min	668
Operating speed No. 2 motor-generator set, r.p.m. max	722
min	668
Voltage No. 1 generator, east skip	535
west skip	535
Voltage No. 2 generator, east skip	535
west skip	535
D.C. line amperes, max., east skip	3900
west skip	4000
D.C. line amperes, sustained, east skip	1300
west skip	1350
Exciter voltage, east skip	247
west skip	247
A.C. line voltage, start	6700
stop	6700
A.C. frequency, start	25.00
stop	24.75
Power consumption, kw.-hr	1050
Power swing, max./min	See Chart
Number of skips hoisted	95
Average time per trip, seconds	38.5
Rope speed, r.p.m., average	2360
Kilowatt-hours per skip	10.05
Over-all efficiency, per cent. based on 10.5 tons per trip, 811 ft. lift	56.2

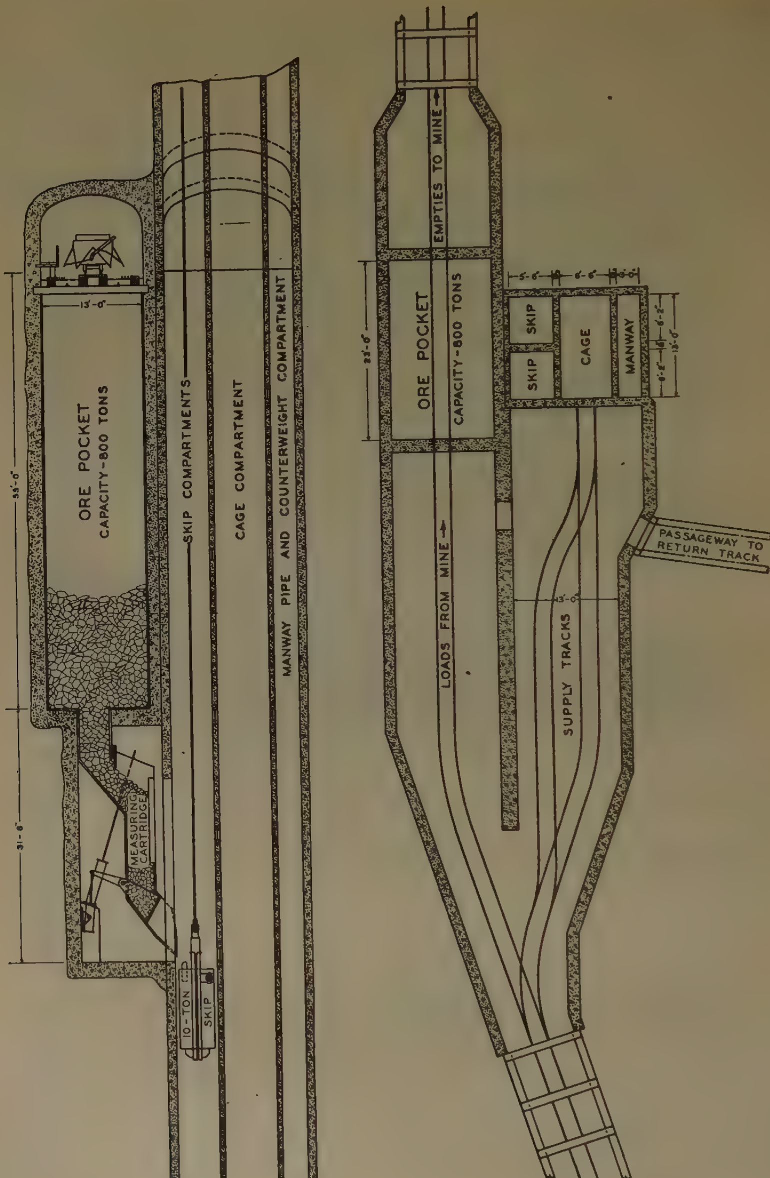

Fig. 16.—Typical shaft station.

the hoist was equipped with automatic control operated through a push button by the skiptender at the underground loading station. This control makes possible hand operation by engineers on the hoist platform, if desired.

Table 6 gives operating data obtained during a test run of this new equipment with the hoist operating under automatic control.

Men, supplies and waste rock are handled on a single-deck counterbalanced cage operating in a compartment 6 ft. 6 in. by 13 ft. This cage has a capacity for handling 60 men per trip; the deck is long enough to handle most of the mine supplies on cars without reloading and most of the underground equipment, including 6-ton trolley and storage-battery locomotives. Fig. 16 shows in plan and section one of the shaft stations and ore pockets.

Underground hoisting between levels is handled by one large, warehouse-type elevator, with single deck large enough to handle all mine supplies and equipment without transferring. Men are handled underground between levels not connected with main shaft by an automatic elevator with push-button control, similar to those used in apartment houses.

Ventilation

Primary ventilation is provided for the mine through shafts 3, 4 and 5 (Fig. 17). Shafts 3 and 5, which are both concreted, the latter being the main working shaft, are downcast, and shaft 4, which is timbered, is upcast. This ventilation is maintained by a suction fan at the collar of shaft 4, handling 130,000 cu. ft. of air per minute, and a blowing fan at shaft 3 handling 60,000 cu. ft. of air per minute. This combination creates a down draft in the working shaft 5 of 70,000 cu. ft. of air per minute. Of this total amount of air, 90,000 cu. ft. per minute is distributed to the 620-ft. grizzly level and the sublevels above it, while the remaining 40,000 cu. ft. is distributed to the 720-ft. haulage level and levels below it. This volume of air provides 300 cu. ft. of air per minute per man underground on the largest shift. In addition to this primary ventilation, secondary ventilation is maintained where necessary through the use of electrically driven fans delivering 5000 cu. ft. of air per minute and compressed-air fans delivering 2000 cu. ft. per minute, blowing air to isolated workings through flexible tubing 12 and 8 in. dia. As the mine workings are shallow, the temperature of the mine is temperate at all seasons of the year. In the event of deep workings in this district, high temperatures on the lower levels would have to be contended with, as the rock temperature increases 1.5° F. per 100 ft. of depth. This condition was apparent when carrying on exploration work several years ago on the 2600-ft. level, where the rock temperature was 103°.

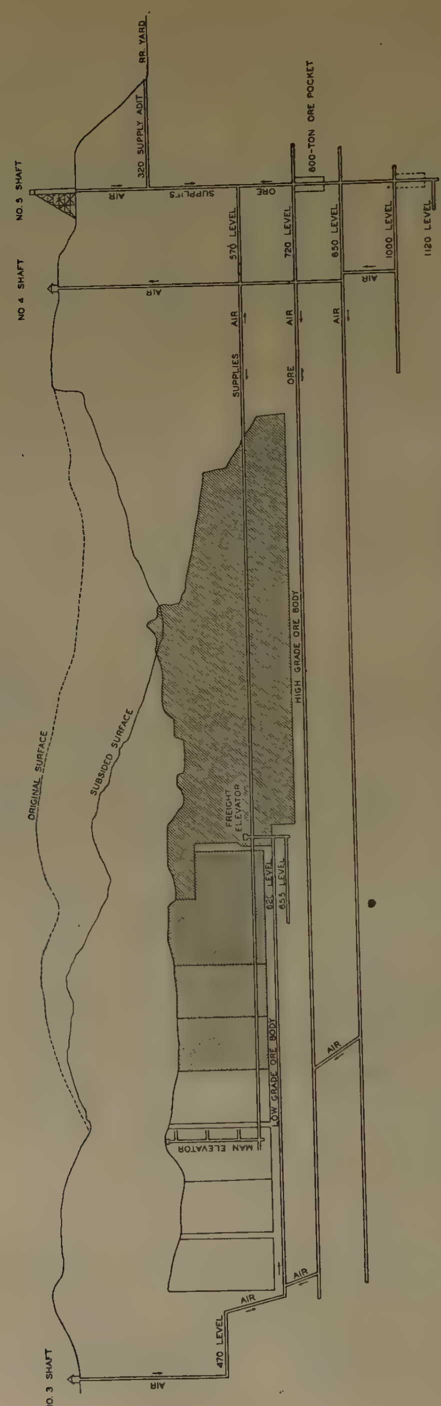

FIG. 17.—LONGITUDINAL PROJECTION OF PRINCIPAL WORKINGS ON E. 250 SECTION.

Drainage

The question of drainage is a simple one, as the mine only makes 70 gal. of water per minute. In this connection the principal effort is directed to draining and ventilating the orebody thoroughly in order to dry it out as much as possible before ore-drawing operations are commenced. The ore as broken and hoisted to the surface contains 3.5 per cent. moisture.

Wage System and Contracts

Wages are to a large extent based on the price of copper, and affected somewhat by the cost of living. The rate for muckers is taken as the standard rate. At the beginning of operations in 1911 this rate was $3.50 and varied between a maximum of $3.75 and a minimum of $3.40. In 1914 a sliding scale based on the price of copper was put into effect, and this rate varied each month with the varying price of copper. This was continued for approximately three years, during which time it went as high as $5.90 in 1917, when copper reached a price of 32c. per pound. Later in this year, when the mines were closed down by labor troubles, and through agreement with the President's Mediation Commission, the sliding scale was abandoned, and a wage of $5.15 fixed for muckers with the price of copper at 23.5c. From this point to the present time, the wage scale has been adjusted 16 times, to keep in step with the varying changes in the price of copper. During the present year there have been five changes in the wage scale, owing to fluctuations in the price of the metal, and at the present time the rate for muckers is $5.06.

About 45 per cent. of all underground labor is on contract. The contracts are let to individuals rather than to contractors employing their own men. About half the contracts are let to men in pairs, and the remainder to men in groups who share equally in the returns from the contract. A number of the contracts are for short jobs, or piece work, such as grizzly raises, chute sets, finger raises, etc., and practically all of the development work is done on contract, which involves longer jobs. The contracts with respect to stoping operations are let by stope engineers, who set the rates and keep records of the work done. Development contracts are let by the development engineers and are measured up by the mine surveyors. Table 7 shows contract rates for labor only, distribution of men to various jobs, and rates of progress for the different operations.

Owing to the multiplicity of short drifts and raises, several work places may be assigned to one contractor, and it is not unusual for him to drill and blast four rounds per shift.

In addition to the jobs listed above, the haulage of ore is on contract at so much per ton of ore hauled to the shaft for labor, and the total

amount is divided equally among the men engaged in this work. Ore drawing is not contracted, owing to the temptation to disregard drawing orders in an effort to get tonnage regardless of admixture of waste rock. Day's wages, according to the scale prevailing at the time, are guaranteed to all workers, and of the total number of men on contract, about 76 per cent. earn

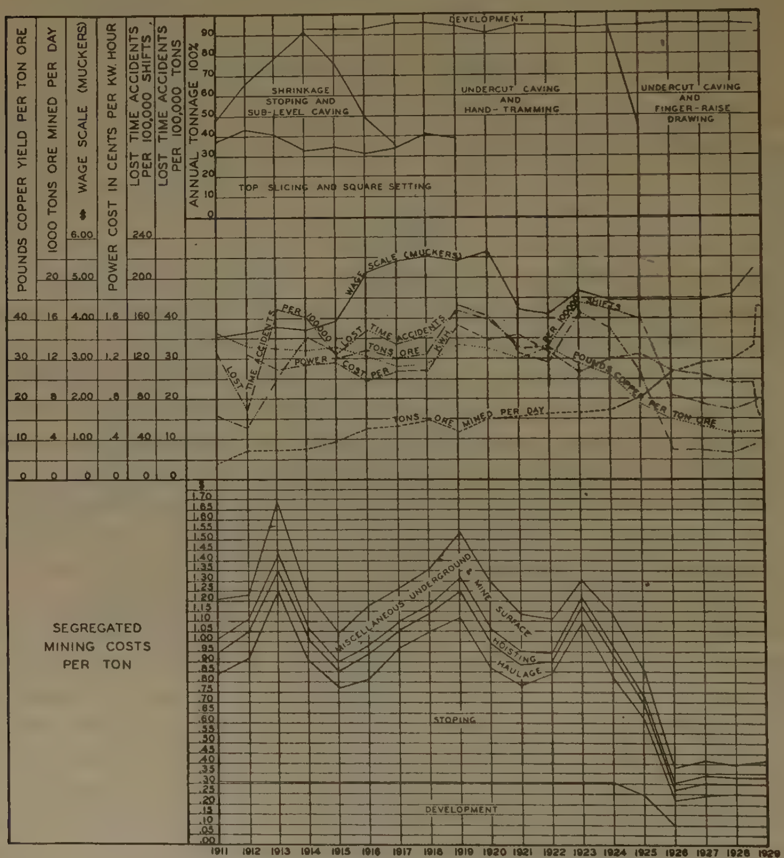

FIG. 18.—HISTORICAL CHART, MINE DEPARTMENT.

more than guaranteed wages. The average earnings per shift of all contracts for the first 10 months of this year have been $1.75 over day's pay. The variation in the wage scale from the beginning of operations to date is shown in the historical chart, Fig. 18. In Fig. 19 is a graph which shows the variations in nationalities of men employed under ground since 1914, indicating an increase in the number of Mexicans employed up to 1923, with a falling off in the number of Europeans, and from 1923 on to the

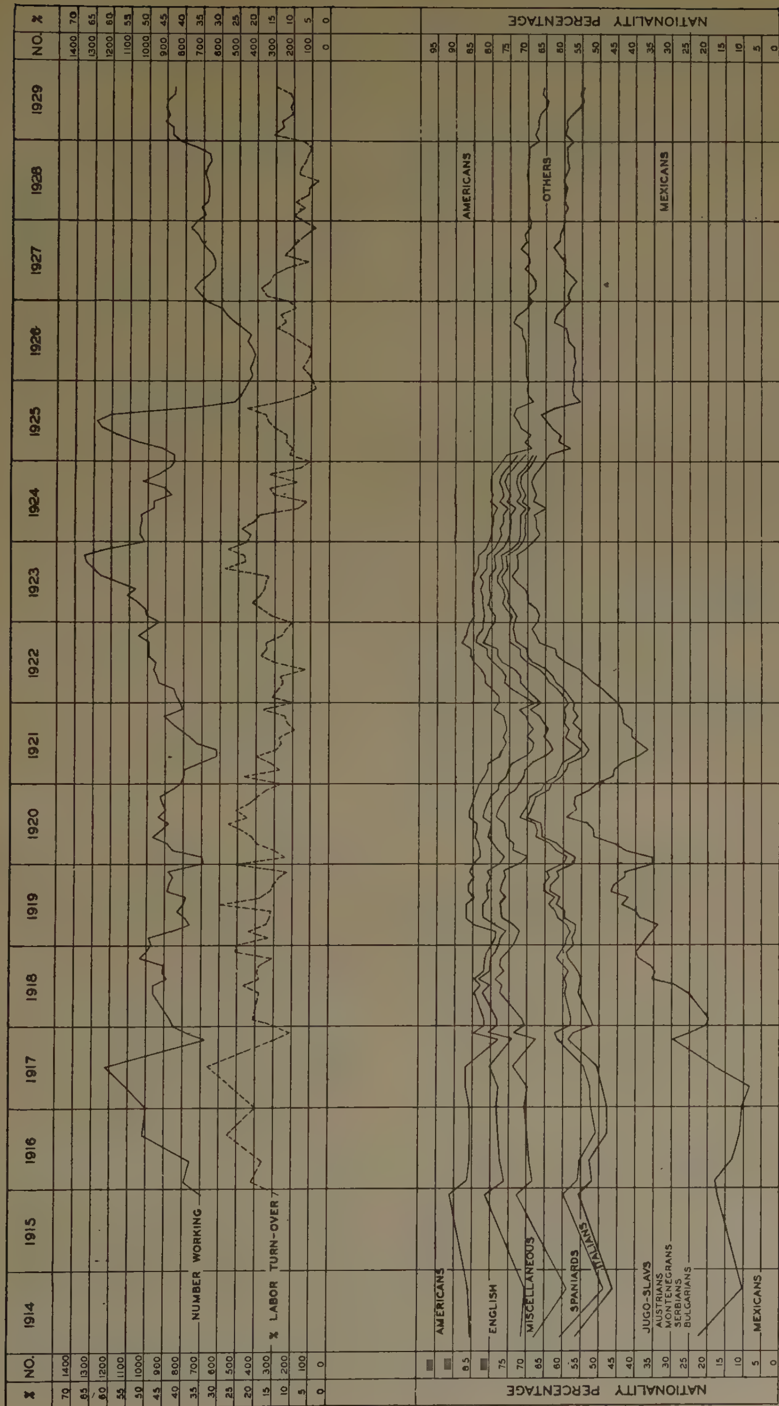

FIG. 19.—EMPLOYMENT RECORD, MINE DEPARTMENT.

present time a falling off in the number of Mexicans and a corresponding increase in the number of Americans.

TABLE 7.—*Distribution of Development and Undercutting Crews with Average Performance and Contract Rates*

Kind of Work	Crew	Man Shifts Required Per Unit	Feet Advance Per Man Shift	Contract Rate for Labor Only	Total Number of Men Assigned to Operation
Haulage drifts	1 machineman 2 muckers		0.89	$7.75 and $10.50	Variable
Transfer raises (major portion cribbed and ironed)	2 machinemen		1.95	$1.65–5.25 per ft.	20
Grizzly drifts	1 machineman		2.267	$1.75–3.95 per ft.	6 to 8
Grizzlies	1 timberman 1 machineman	2		$11.00–13.00 per griz.	2
Grizzly raises (24 ft.)	1 machineman	2.679		$17.00 per raise	2 to 4
Chute-set sills	1 timberman 1 machineman	2.222		$20.00 per sill	2 to 4
Chute sets	1 timberman 1 machineman	7.383		$42.00–52.00 per set	8
Finger raises (13 ft.)	1 machineman	1.315		$10.00 per raise	6 to 8
Undercutting-level drifts	1 machineman		5.465	$1.10–1.25 per ft.	3 to 4
Undercutting-level mining	4 machinemen	410 per stope		$0.14 per sq. ft.	16
Boundary caving drifts	1 machineman 1 mucker		1.94	$2.00–4.50 per ft.	12
Drilling and blasting boundary caving drifts	1 machineman		18.00	$0.33⅓ per ft.	4

MINE SAFETY

The promotion of mine safety is rated as one of the company activities of major importance. In conjunction with two other principal operating companies of this district, a Mine Rescue and First Aid Association was organized in 1918, and a station centrally located for the district was built and fully equipped at a cost of $50,000. One important feature of this equipment is a specially built truck which carries all the equipment necessary to combat a mine fire and to carry on rescue work, and is held ready at all times to answer immediately a call from any of the district mines.

This station is under the supervision of a competent director who has under him three station attendants, one on each shift. The cost of maintaining and operating this station amounts to $0.013 per man shift, charged to each company on the basis of the number of shifts worked. The personnel of this station gives first aid training and mine rescue instruction to the employees of the contributing companies. Each company maintains three mine rescue teams of five men each, and in addition requires all foremen and bosses to be familiar with the use

of all mine rescue apparatus and to review this training once each year. First aid training is not compulsory, but a large proportion of the men take the training. In order to encourage this, the companies offer cash prizes each year for the best teams in several contests held annually. Since the organization of the Association there have been 77 of these contests held, with 291 teams competing, and $27,280 has been paid out in prize money.

In Fig. 18 the lost time accidents per 100,000 shifts and per 100,000 tons are shown. The former indicates a drop from a maximum of 174 in 1923 to 69 in 1928, with a corresponding drop in the accidents per 100,000 tons of ore mined from 42 to 7. The highest accident rate, which was during the year 1923, was at a time when all of the ore was being mined by undercut caving and hand tramming. A previous high peak in the accident rate was in 1919, when about 39 per cent. of the ore was mined by top slicing and 55 per cent. by undercut caving and hand tramming, the remainder being development ore, and the highest rate per 100,000 tons mined was during 1919 when a large proportion of the ore was mined by top slicing. A comparison made some years ago, before top slicing was abandoned, indicated that in this method of mining, while the accidents per 100,000 shifts were lower than in caving, the accidents per 100,000 tons mined were higher, owing to using a greater number of men with this method. The present mining method has made a good accident record by comparison with previous methods in use, both on the basis of shifts worked and tons mined.

The greatest contribution a mining company can make toward mine safety is to plan the mining method and the details of operations connected with it in such a way that it is inherently a safe rather than an unsafe operation.

Fire Protection

In addition to the apparatus available at the mine rescue station for fire fighting, emergency apparatus mounted on a 24-in. gage mine truck is stationed at the collar of the shaft ready for instant use. This truck is equipped with six ½-hr. oxygen appliances, with extra oxygen cylinders and regenerators, four miner's self-rescuers, one pulmotor, flashlights and miscellaneous tools and supplies. Underground stretcher and first aid stations are located throughout the mine. Timbered shaft and winze stations are equipped with high-pressure water hose, while timber over underground electrical installations is protected with automatic sprinklers and Pyrene or Phister hand extinguishers. Three hundred soda-acid 2-gal. extinguishers are distributed throughout the timbered section of the mine and Foamite 5-gal. extinguishers are placed near hazards from oil or methane gas. At strategic points underground are kept gas masks of the Burrell all-service type and miners' self-res-

cuers. For the treatment of minor gas cases arising from powder gases, oxygen cylinders equipped with inhalator masks are available.

Mining Costs

During the 4-year period from Oct. 1, 1925 to Sept. 30, 1929, inclusive, 16,556,296 tons have been mined by the method now in use; costs per ton for this are shown in Table 8. The average daily tons mined during

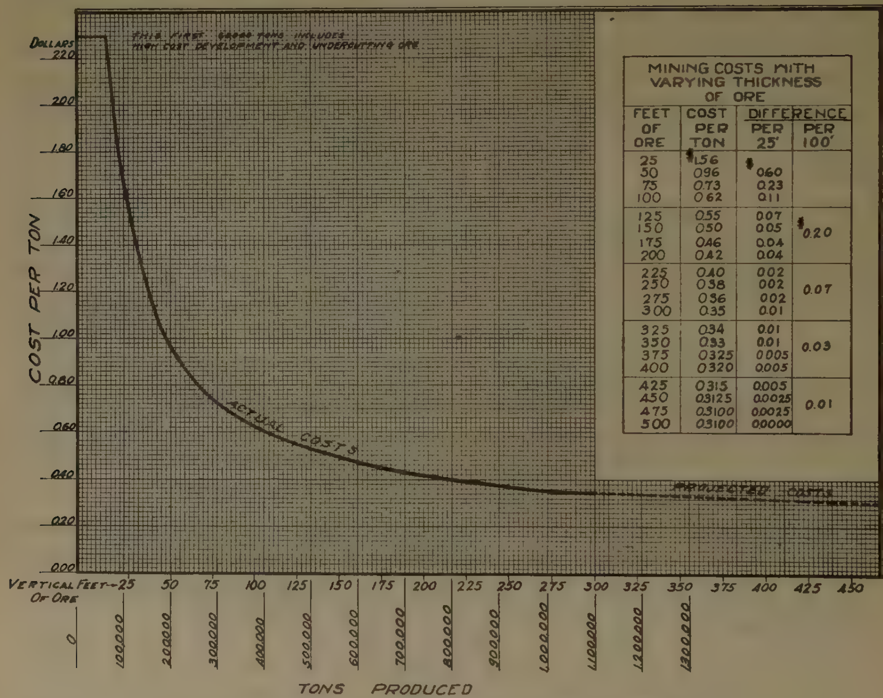

Fig. 20.—Mining costs, Miami Copper Company.

this period was 11,800. This has since been increased to 18,000 tons daily.

Table 8.—*Mining Costs during Four-year Period*

Oct. 1, 1925 to Sept. 30, 1929. Tons mined, 16,556,296

	Cost per Ton
Development	$0.10000
Stoping	0.13628
Electric haulage	0.05314
Hoisting	0.03317
Ventilation, etc	0.02082

	Cost per Ton
General underground	0.01149
Engineering and sampling	0.01382
Mine surface	0.01979
Mine accident	0.01086
Total	$0.39937

In Fig. 20 appears a graph of mining costs which shows the cumulative cost per ton of ore mined as a stope progresses. All of the development

and undercutting are charged out against the first tonnage, bringing the cost of the first 60,000 tons or so up to $2.30. As the tonnage derived from ore drawing at its comparatively low cost is combined with this high preliminary tonnage cost as the stope progresses, it brings the cost curve down rapidly to begin with and more slowly during the later stages of the stope. From this curve may be taken the cost of mining by this method for any given thickness of ore. It also indicates that while the costs come down rapidly if the thickness of ore is increased up to a certain point, the curve flattens out and does not show much advantage in the cost after a thickness of 300 ft. has been reached; so that if one had to deal with an orebody 600 ft. thick it would probably be best to mine it in two lifts of 300 ft. each, rather than in one lift of 600 ft., particularly if the area were large and required to be mined in several units rather than in one operation.

Capital Expenditure for Increased Tonnage

When it was decided to mine the low-grade orebody it was at the same time decided to increase the tonnage capacity of the plant. During the last year of operations on the high-grade ore, the tonnage handled was 6800 daily. Since then the capacity has been increased to 17,000 tons daily for the mill and 18,000 tons daily for the remainder of the plant, at a capital expenditure shown in Table 9.

Table 9.—*Capital Expenditure*

	Total Expenditure	Increase in Daily Capacity, Tons	Cost per Ton
Mining plant	$ 371,810.37	11,200	$ 33.20
Milling plant	2,303,617.95	10,200	225.84
Power plant	1,235,693.93	11,200	110.33
Water development	27,911.86	11,200	2.49
Total	$3,939,034.11		$371.86

The addition to the power plant was of sufficient capacity to produce all the power required for the entire tonnage rather than for the increased tonnage alone. This consisted of a 15,650-kva. turbogenerator unit with boiler plant operating at 400 lb. steam pressure, with the old 200-lb. pressure equipment held in reserve. The new equipment produces power at an operating cost of 6 mills, with $2.00 fuel oil compared with 10 mills per kilowatt-hour with the old low-pressure equipment.

Miscellaneous

The average number of tons mined per shift underground over the 4-year period was approximately 27.

Timber consumed per ton of ore mined amounted to 1.045 bd. ft. All timber that is expected to remain in the mine two years or more is treated in the company's pressure treatment plant. In 1928, the plant treated 819,796 bd. ft. of timber at a cost of $6.85 per 1000 bd. ft., the average retention being 0.62 lb. of zinc chloride per cubic foot of timber treated.

The explosive used is 40 per cent. gelatin powder. The consumption is 0.2225 lb. per ton of ore; 4.5 tons of ore are produced per pound of powder. The consumption of powder in the operation of ore drawing alone, consisting principally of chute blasting, is such that 19.7 tons of ore are drawn per pound of powder used.

The power required in the mining department per ton of ore mined is approximately 1.9 kw-hr. Of this amount, slightly more than 1 kw-hr. per ton is required for hoisting.

Among other things shown on the historical chart of the mine department (Fig. 18) is a marked increase in the number of tons of ore mined per day, which has increased from a minimum of 2851 tons per day in 1912 to 18,000 tons per day at present and a corresponding decrease in the pounds of copper yield per ton of ore from a maximum of 33.21 in 1912 to 11.68 lb. per ton for the year 1928. The combination of these two results in an annual copper production at present slightly higher than the highest previous production of the company, and while it is not shown on the chart it is interesting to note that the mill tailings during the first three years operation of the property contained an average of 14.52 lb. of copper per ton. In other words, the ore being mined and treated today yields less copper per ton than was contained in the discard of 15 years ago.

DISCUSSION

H. G. Moulton, New York, N. Y.—Largely as the result of the operations under Mr. Maclennan's direction at Miami, much copper ore is now within the range of commercial extraction by underground mining methods, which was considered as waste six years ago. Attention is called to several important features of this paper, which are of permanent value for the notebooks of all engineers having to decide on the possibilities of other low-grade copper deposits. As an illustration, the paper contains some exact figures on dilution and on the effect of varying thicknesses of the orebodies on the mining costs. We are indebted to Mr. Maclennan and to the officers and directors of the Miami Copper Co. for their generosity in releasing so much valuable information for the benefit of the profession as a whole.

It is apparent that a large measure of the success of the mining operation as conducted at Miami, both from the standpoint of recoveries and of costs, has been due to the extensive and careful engineering supervision of all details of underground operation.

As this is a joint meeting of the Committee on Mining Methods and the Committee on Ground Movement and Subsidence, of both of which Mr. Maclennan has been so active a member, the Committee would appreciate some comments on the

ground movement and subsidence problems involved in connection with the Miami operations.

F. W. Maclennan.—Answering the chairman's request for comments on the ground movement and subsidence problems involved in connection with the Miami operations, reference is made to a paper[3] presented in February, 1929. This paper includes plans, sections and photographs, and goes into the subject of subsidence in greater detail than time would permit of doing here.

In its practical application to mining operations the most important information gathered regarding subsidence is the fact that the ground when undercut over comparatively small areas, as detailed in the paper, will cave and break up through the effect of its own weight and movement and that once this ground is broken up it will draw down vertically through comparatively narrow pipes without much lateral spread. This characteristic necessitates close spacing of the chutes but avoids dilution of the ore drawn in the pillar stopes through lateral movement into these stopes of waste filling in the original stopes.

C. E. Arnold, New York, N. Y.—Undoubtedly Mr. Maclennan's paper is an exceedingly valuable contribution to mining literature, as it summarizes a vast amount of valuable experience in undercut caving. In my opinion, any man confronted with the problem of starting a caving operation can, by application of the principles developed by Mr. Maclennan, save himself a great deal of trouble and money.

At the bottom of page 65 there appears the statement: "The following statistics represent the average results of drawing the 13 complete stopes, producing 12,710,378 tons of ore, of which 79.59 per cent. was drawn clean; that is, with no dilution from capping." Does this statement mean that in charting, say, from month to month, the cumulative ore tonnage drawn and the cumulative copper tonnage recovered therein, the two chart lines would not diverge until arrival at the point where 79 per cent. of the tonnage had been drawn?

F. W. Maclennan.—The statement means that 79.5 per cent. is drawn clean. This figure is obtained by totaling the number of tons that are drawn clean before waste has appeared in each individual chute and dividing this amount by the estimated amount of ore in place. The percentage drawn that would apply to the chart you refer to would be about 25 per cent.; that is, dilution by capping which has begun to appear at some point in the stope, when approximately 25 per cent. of the total stope tonnage has been drawn, but clean ore continues to be drawn from other parts of the stope for a long time after the first capping appears and this continues up to 79.59 per cent. as detailed above. Does this answer your question?

C. E. Arnold.—Yes, surely. I should like to say that Mr. Maclennan's paper can be seen in far better perspective by reading it in conjunction with the annual reports that his company publishes. These reports show what a remarkable effect the mining performance has on the economics of the company's whole operation.

G. S. Rice, Washington, D. C.—When I was last at the mine, about three years ago, before any of the pillars had been drawn, one of the problems was to what extent there might be a caving in of the thin curtain pillars between the adjacent stopes and so a drawing in of the waste from the previous filling.

I notice you say there was some dilution. I wonder if that dilution came through from above, or whether it came from the adjacent stope which has been filled.

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[3] F. W. Maclennan: Subsidence from Block Caving at Miami Mine, Arizona. *Trans.* A. I. M. E. (1929) 167.

Another question, relating to the piston-like action of the orebody in coming down in the stope: Is the effect under control by the spacing of your waste drifts? I note you have changed the spacing from 30 ft. to 45 feet.

F. W. Maclennan.—I will answer the last question first. When we originally spaced the boundary caving drifts at 30-ft. vertical intervals we contemplated weakening the stope boundaries 25 per cent., which this spacing does, as it removes 7½ ft. of ground from 30 ft. Later on we changed this vertical spacing from 30 to 45 ft. for the sole purpose of economy and not for the purpose of improving control, as drifts can be driven and the backs shot in at 45-ft. vertical intervals for less money than they can be driven at 30-ft. intervals without shooting the backs in. This 45-ft. arrangement weakens the boundary 33⅓ per cent., as it removes 15 ft. of ground from 45. We have no data to indicate whether a better control of caving results from either arrangement.

Regarding the other matter of dilution of the pillar stope from the column of waste adjoining it: The necessity for drawing ore which adjoins a column of waste is not new with this method. The same condition existed in our old caving method in which the ore was divided into panels 150 ft. wide and extending across the entire width of the orebody. In this panel method three or four panels 150 ft. wide would be started from one side of the orebody, leaving 150-ft. pillars between, and a year or so later, when the column of broken capping, which had caved down into the original panels, had become consolidated and the original panel ore had been drawn back several hundred feet from the side of the orebody, drawing was started in the pillar panels from this side and they were drawn back between the columns of consolidated broken waste lying in the original panels. This work indicated that it was desirable to leave a thin partition or pillar of ore to avoid dilution of the ore drawn in the pillar panel by waste coming in laterally.

It was usually advisable to avoid drawing the side chutes in the pillar panel for a while until caving was well started, when they could be drawn with less liability to dilution. This experience led to the leaving of thin partitions between stopes in the present method and the records indicate that they have been effective in preventing much lateral dilution; the records also tend to indicate that we are drawing some of the ore in these partitions.

In drawing these pillar stopes it is frequently noted that the first waste to appear comes from the center of the stope rather than from the sides where it would be expected. In a recent stope it started in several chutes right in the middle of the stope, but I think on the whole there can be no doubt that the ore drawn from the sides of the pillar stopes is more diluted than the general average of the stope. We have no definite figures on this, for the reason that our samples are taken at the haulage-level chutes, and the raise from these chutes branches in such a manner that it is impossible to segregate the ore from the chutes along the border of the stope.

Originally we intended to put boundary caving drifts around the pillars as well as around the original stopes, but when we put the first drift along the side of the first pillar stope, No. 9, 15 ft. from the edge of the pillar, the ore along the side of the pillar was so shattered that the drift required timbering to hold it open and it seemed a waste of money to weaken this boundary any more than it already was weakened by the mining of the adjoining stope. Crosscuts from these pillar boundary caving drifts were put in the waste fill in adjoining stopes, Nos. 1 and 2. The fill in these original stopes seemed to be about as compact and solid as the ore in the adjoining pillar and it appeared that once this ore was undercut it would draw down vertically with very little spread. I remember on one occasion the ore was bounded by a wall which dipped at about 70° to the vertical, so that instead of cutting off the stope along a vertical boundary line it was cut off along a 70° plane, hoping that the draw would

spread. Considerable of the ore lying on this steep footwall might be drawn into the chutes along the boundary but the records indicate that the draw was vertical and that none of this ore beyond the vertical plane, although it had been cut off, was drawn into the chutes. I think one's mental attitude is apt to be such that if he wants the draw to spread from the vertical and bring in additional ore he is afraid it won't, and if he doesn't want it to spread from the vertical and bring in waste dilution he is afraid it will, but fortunately for this system of mining, the spread from the vertical is very little in either case.

R. D. Hoffman, New York, N. Y.—There is one point regarding Miami which has not been considered; namely, that the mine was a going one at the time the low-grade ore was considered, and practically all of its capital charges had been paid out when this new operation started, hence there was not the serious consideration of amortization of capital. I would not like to add too much work for Mr. Maclennan, but I wonder whether he would be good enough some day to compile a set of tables indicating what Miami's costs would have been if the mine had been compelled to erect a new plant and start a virgin operation. Those of us in the field who occasionally examine a low-grade orebody would consider such a compilation as our Bible, taking into consideration, however, that just as there is only one Calumet & Hecla mine in Michigan after 100 years of mining, there will probably be only one Mr. Maclennan to handle the problem of caving very low-grade ores.

C. H. Crane, New York, N. Y.—I had the pleasure of visiting Mr. Maclennan last fall. I went to see how a caving operation could be done. The thing that impressed me most was not the beautiful sight of the caving operation, but the teamwork, the spirit of the Miami crowd, with Mr. Maclennan at the head of it. It is true that the mining is being handled by engineers, but by engineers who absolutely trust the man at the head; and without that trust and that complete cooperation I do not think they could do what they have done. I should not advise taking this as an example and going out into the wilds expecting to equal it.

H. G. Moulton.—Mr. Maclennan, at the place where your caving takes place, at the top of the finger raises where you drift across and drill, how much of that area expressed in percentage has to be mined out? In other words, for every ton brought out of the mine, how much of it had to be mined and how much of it caved? I do not mean to include there all of the tonnage coming from development work, but just the actual shooting out, to induce caving.

F. W. Maclennan.—I am sorry I did not include that figure in the paper. I would say about 4 per cent. This and similar figures I think can be arrived at by computation from plans and data appearing in the paper.

H. G. Moulton.—Also applying that figure to it on the caving level itself, where the caving takes place, a certain amount of that is taken out with drifts run through, and the rest of it comes from caving after you have shot the supports. What percentage of that area, included in the height of a drift, is actually mined and trammed out of drifts? Have you a figure in mind?

F. W. Maclennan.—The mining level area that is undercut is 150 ft. square. It has from six to eight small primary drifts 4 ft. wide, depending on the character of the ground, and in addition to this, six secondary drifts 8 ft. wide, from which the pillars are drilled. The broken ground is all cleaned out of these drifts but no tramming is required on account of the close spacing of the finger raises. This drift area amounts to from 48 to 53 per cent. of the total area undercut, depending on whether six or eight of the original small drifts are put in. It is not only advisable to muck out

these drifts to facilitate setting up to drill the pillars, but it is advisable to do so in order to accommodate the swell of the pillar ground when this has been blasted in. Otherwise it is liable to become packed by the blasting and made difficult to draw.

A. Notman, New York, N. Y.—I'd like to consider whether the Miami Copper Co. has maintained its proportion of the business, in spite of the fact that it has had to mine material of so much lower grade. I have prepared some figures that show that in a group of companies, the larger copper companies, who in 1915 were producing 60-odd per cent. of the world's output, have maintained, in fact, gained a little on that proportion at the present time, and have somewhat more than doubled their output of copper per annum.

The figures Mr. Maclennan gives in the last paragraph indicate about the same ratio; that is, in 1912 with recovery of something over 33 lb. per ton, they were producing about 95,000 lb. of copper per day; and with this present recovery and present tonnage, they are producing about 198,000 lb. of copper per day, showing that they, too, have maintained this same proportion of the total business, in spite of the handicaps of lower grade.

T. T. Read, New York, N. Y.—When 12 lb. was selected as the limit, was it an economic choice or a natural one? Could you perhaps have developed 120,000,000 tons with an 11-lb. cut-off, but the 12-lb. cut-off gave you the greatest net return over the whole job? In other words, it is a sort of budget figure, is it not?

F. W. Maclennan.—The 0.6 per cent. cut-off figure is primarily an economic one, although it also happens to be a natural one, as the grade drops sharply from this point. With a cut-off at this grade a reasonable profit can be made on the invested capital. This 0.6 per cent. is the cut-off of the ore in place, and a much lower one, of course, is used in our ore-drawing operations after the ore has been caved and ready for ore drawing and this latter cut-off grade varies, depending on the cost of operation in the particular stope, market price of the metal, total profit requirements and other factors.

R. Peele, New York, N. Y.—Figures are given from which could be derived the approximate length of life of the mine, but of course, future improvements, leading to the mining of a still lower grade material and further discoveries, might alter the computation made on the basis of these figures. Does Mr. Maclennan wish to make any statement with regard to the probable life of the mine?

F. W. Maclennan.—On the basis of our present tonnage, and the ore which we include in our ore reserves, the present life of the mine is about 16 years. There is an old saying, though, that a good mine dies hard. Personally, I think it will have a considerably longer life.

C. F. Jackson, Washington, D. C.—It seems to me, in reading between the lines, there is one other thought that is well worth mentioning; that along with the exceptional skill in solving the purely technical problems, and in the planning, there has been an exceptional skill in the carrying out those plans and the coordinating of the various operations making up the general underground operation.

The coordination of the work is well illustrated in Table 1. This is perhaps one of the simpler phases of the operation to coordinate, but I was struck by the fact that the work started on time and was finished on schedule.

Member.—I was wondering what the milling difficulties might be in such a case. As I get it, the ore begins to be diluted at 25 per cent. on some of the withdrawals, and if it averages over the entire 79 per cent. of content, and you begin with something that is nine-tenths and end with something about three-tenths, the average of which

would be six-tenths, I am wondering whether the recovery in the mill would be constant, and what effect that irregularity has on the mill operation.

F. W. MACLENNAN.—It is true that there is a great variation in the grade of the ore coming from an individual stope between the beginning and the finish of drawing this stope, but as the output of the mine is derived from 13 to 14 stopes in different stages of ore withdrawal, and as the ore is hauled from all of these stopes to the shaft continuously, it is rather thoroughly mixed in the shaft pocket and during hoisting and coarse-crushing operations, and when it arrives in the mill storage bins it is of fairly uniform grade. The extraction in the mill is improving steadily due to refinements in the process and at the same time the cost of milling has been brought down to less than 29 c. per ton, including coarse crushing.

J. P. HODGSON, Bisbee, Ariz. (written discussion).—Since reading Mr. Maclennan's paper, the writer has had the pleasure of a trip to the property to investigate some of the details connected particularly with the mining of the ores as described by Mr. Maclennan. On arrival one is struck by the neat and clean appearance of the surface plants, indicative of good housekeeping. The mine superintendent's department, engineering staff, etc., are all housed in a building that is close to the main operating shaft, where they are easily kept closely in touch with each other and with the operations.

The hoisting plant, which is handling at present approximately 18,000 tons per day, has been remodeled and equipped with an additional motor and generator set. The rope speed has been increased to 2250 ft. per minute, operating two 10-ton skips in balance, and by these changes it has been made comparatively easy to hoist 90 skips per hour, thus giving the production needed in approximately 20 hr. of hoisting time—a very efficient operation.

The entire loading and hoisting operations are controlled by one man, who handles the mechanical apparatus for loading the skips, and by push-button control also operates the hoisting engine. This man is housed in a small dustproof building, or cabin, with windows so that he can easily see the loading apparatus and the skips as they are being loaded and dispatched. The skips are loaded from a pocket of 800 tons capacity, through which passes all of the ore from the mine.

The haulage serving the entire mine comprises 6-ton locomotives in tandem, hauling trains of 35 cars, each car with a capacity of 3.6 tons. These trains are unloaded at the shaft while in motion, the average time of unloading a train of cars being from 3 to 6 min. The main haulage tracks are laid with 65-lb. steel rails and the haulage drifts under the orebody are spaced 150 ft. apart. The entire operation is in charge of a train dispatcher with telephonic communication to various parts of the level, thus giving him entire control.

The present Miami block-caving system has been in operation approximately four years. The property prior to the installation of this system had an admittedly short life, due to the extremely low copper tenor of the ore. The idea of caving blocks of such low-grade material up to a total height of 350 ft. had not, up to that time, been visualized and put into active operation, so that the Miami management undoubtedly first envisaged and took advantage of the possibilities of such an operation.

Mr. Maclennan and his staff realized that successful operation could be possible only by obtaining a phenomenally low mining cost and a low beneficiating cost at both mill and smelter. The mining problem was the most important, since, due to the chemical constituents of the ore and its adaptability to high ratios in milling practice (approximately 40 into 1), a concentrate of about 36 per cent. copper was obtainable, this in turn resulting in a relatively low smelting cost per pound of copper.

Mr. Maclennan described very clearly in his article the plan of the haulage raises, grizzly drifts, finger raises, undercutting drifts, etc., so that it is unnecessary for me

to make any comments upon these except to say that for the type of ore, the layout is admirably adapted.

If the ore broke coarser than it does, the scheme as laid out at Miami could not give the results which are being obtained, but in a measure nature has compensated for the very low-grade material by making it break easily. It is friable and relatively fine, and is drawn off easily through finger raises and grizzlies into the haulage chutes, permitting quick loading into cars, and, as the ore contains little moisture, these conditions become valuable and favorable factors in the mining cost.

According to the information given as to dilution, it would seem that to date this part of the problem has been held within reasonable limits. One reason, I believe, that makes this possible is that the 15-ft. vertical barrier pillars surrounding each stope are of material assistance in holding back waste from adjoining worked-out sections, so that a minimum of side dilution occurs. However, it is noted that these barrier pillars were eliminated in the original calculations of estimated minable ore, leaving only 89 per cent. of total ore presumably to be mined. In practice, therefore, an unknown quantity of this ore is undoubtedly mined, so that the actual dilution is probably considerably higher than 10 per cent., notwithstanding the fact that the waste itself packs readily in the stope sections after their completion.

I feel that the Miami Copper Co., in the planning and successful extraction of an exceedingly low-grade orebody, as indicated by the remarkably low costs obtained, has taken a decided step forward in mining practice, and in a measure has pointed the way to others who may have similar conditions.

Vertical and Incline Shaft Sinking at North Star Mine

By Arthur B. Foote,* Grass Valley, Calif.

(San Francisco Meeting, October, 1929)

At the end of the year 1914, the main North Star incline shaft had reached the 6300-ft. level, and encountered a vein dipping southwest, or exactly opposite to the North Star. Subsequent development failed to find the North Star vein continuing beyond this intersection, so the 6300-ft. level proved to be the bottom level of the North Star vein, and very little ore was found below the 5300-ft. level. The new vein was called No. 1 vein. Development work carried on for the next 10 years discovered a third vein, called No. 2, branching off about 1200-ft. to the east, with a north-south strike and a west dip. No. 1 vein had about 900 ft. of backs above the 6300-ft. level before reaching the boundary line of the property, and No. 2 vein died out about 800 ft. above. During these 10 years, the money spent on development was increased from $54,000 in 1914 to $184,000 in 1924; the ore reserves steadily diminished until it was evident that the mine must close down unless by some means a large tonnage of ore could be developed quickly.

Problems and Plans

In order to understand what follows, the general layout and equipment of the mine as it was at this time must be briefly described. Fig. 6 indicates the relative positions of the shafts.

The North Star (incline) shaft extends from the surface to the 6300-ft. level following the vein, with an average pitch of 26°, between limits of 11° to 38°. There is at present no hoisting equipment in this shaft between the 2700 and 4000-ft. levels. The Central (vertical) shaft connects with the North Star shaft at the 4000-ft. level, at a vertical depth of 1600 ft. In two compartments of this shaft 4-ton skips made a vertical turn of 15 ft. radius at the 4000-ft. level and ran down the incline to the 6300-ft. level. A cage operated in the third compartment to the 4000-ft. level. Men and supplies were transferred to a truck running on a third track down the incline shaft to the 6300, operated by an electric hoist on the 4000-ft. station. When the shifts changed double-deck cages were substituted for the skips in the vertical shaft, but they stopped at the 4000-ft. station. The truck to 6300 carried 22 men, and it required 8 min. to make a trip down and back. This arrangement was satisfactory as long as a fairly large proportion of the work was

* General Manager, North Star Mines Co.

within walking distance of the 4000-ft. level, which was the case for 20 years; but when practically all the men, timber and tools had to transfer at 4000-ft. station and be transported by this single truck to the 6300, the congestion and loss of time became serious. Men could be landed at 4000 station at the rate of 22 men every 3 min., but from there to 6300 the rate was 22 men in 9 minutes.

During the year 1924, the problem was studied, and it was finally decided:

1. To sink an incline shaft on No. 1 vein in line with the main North Star incline shaft, which would follow the vein down until it was vertically under the Central vertical shaft, cutting stations every 300 ft. The skips operating in this new shaft would dump into the loading pockets at the 6300-ft. station of the North Star shaft. If the pitch of the vein continued without change, the incline would have to be sunk 2300 ft. from the 6300-ft. level.

2. To sink an incline shaft on a promising oreshoot on No. 2 vein from a point on the 6000-ft. level, about 2700 ft. southeast of the 6000-ft. station of the North Star shaft and to cut stations every 300 feet.

3. To sink the Central vertical shaft approximately 2000 ft. to connect with the No. 1 shaft.

4. To connect the bottom of No. 2 shaft with the No. 1 and vertical shafts by a drift on the veins. This to be the 8600-ft. level (the distance measured along the dip of the No. 1 and North Star veins to the surface).

5. While this was being done, to continue stoping as much ore as possible, and, in order not to run out of ore, to start drifts from the incline shafts as fast as stations were cut.

It was fully realized that this program would tax the hoisting and compressor capacity to the limit, but to stop production for the time necessary to complete the sinking of the shafts would involve too large a financial outlay, even if it should prove cheaper in the end. This was chiefly due to the large overhead charge, mostly made up of pumping cost and taxes. The time required to complete the program was of great importance. It was estimated that expenses would exceed production by about $1000 a day until the completion of the program.

In a job of this kind, conditions never being exactly the same, some new methods must always be devised. Experiments carried on during sinking operations are extremely expensive; a record of mistakes made may be of as much value as the description of the successful method, therefore the plans that failed and the unforeseen obstacles that had to be overcome will be particularly described.

Deepening the Vertical Shaft

The problem was to deepen Central shaft about 2000 ft. The work must not interrupt continuous hoisting through the shaft. As no vein

was known to exist in the ground to be penetrated, there was no necessity for cutting a station before reaching the bottom. The rock was hard, either diabase or granodiorite. Little water was expected until No. 1 vein was reached, but if a flow should be encountered without provision having been made to handle it, an extremely expensive delay would result.

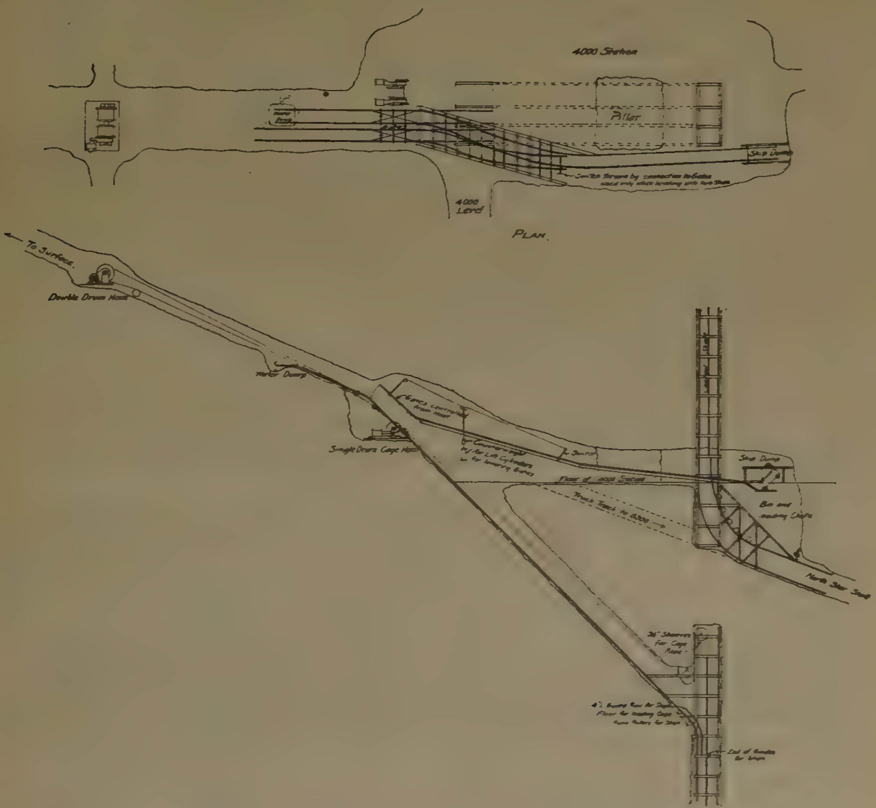

FIG. 1.—RIG FOR DEEPENING CENTRAL SHAFT.

The rig for sinking is shown in Fig. 1. Starting about 60 ft. up the incline shaft, which was 23° at this point, a 45° incline was sunk until it was vertically under Central shaft, and the new shaft started down, leaving about 20 ft. of solid ground between the bottom of the old shaft and the top of the new excavation. The timbers were, of course, carefully lined up so that, when the connection with the old shaft was finally made, the guides and rails would go straight through.

Hoisting

The sinking hoist was located 100 ft. up the North Star shaft. It was a double-drum hoist with band clutches and post brakes, geared to a

100-hp. 440-volt induction motor. Its capacity was 7000 lb. pull at 500 ft. per min. This hoist had been converted in the mine shops from a first-motion cross compound hoist designed for compressed air and originally installed in 1896. In order to facilitate erection underground, it was assembled on a structural steel bedplate. A separate single-drum geared hoist driven by compressed air was connected up to operate a cage in the third compartment of the shaft. This cage was useful for installing the pipe lines, electric cables, etc., and also acted as a stand-by to get the men out in case of a failure of the main hoist. It thus made a ladderway unnecessary.

Each skip compartment required for the operation of the main skips four lines of 20-lb. rail and two 4 by 4-in. wood guides. Only the guides and two of these rails were put in place as the shaft was sunk. The rails were extended up the incline shaft. When hoisting rock, the skip was pulled up the incline shaft some distance above the station. The hoistman then dropped a gate by means of an air-lift, the skip was allowed to run back over the gate and to dump into the loading pockets for the main skips. These pockets were on the far side of the shaft from the sinking hoist, so the track had to curve around the end of the shaft. The tracks from both compartments were switched to one dumping track, the switch being thrown mechanically to connect with the particular gate that was dropped by the hoistman.

The wall plates, end plates and dividers are 8 by 8-in. native spruce with 6 by 6-in. posts. There are two 4 by 5-ft. skip compartments and a 5 by 5-ft. compartment for the dolly cage and pipe lines, etc. The cage leaves a 16-in. space clear on one side for the pipe lines. This makes the shaft 6 ft. 4 in. by 15 ft 8 in. outside the timbers. Sets are spaced 5-ft. centers, and this spacing was made as exact as possible so that guides, etc. could be drilled to template and fit anywhere. Bearing sets were placed every 200 ft. Practically no lagging was used, the rock being hard enough to stand alone.

The incline approach to the top of the shaft made the use of skips for hoisting necessary, and the distance to be hoisted before the job was finished required skips of 2½-ton capacity. It was obvious that these could not be loaded by shoveling into them, both on account of their size and the time that would be required to make a round trip. J. Fred Johnson, who secured the contract for sinking the shaft, proposed the use of loading pans. The difficulty of applying this method in the case of this small shaft and large skip was worked out as follows:

A guide for the wheels of the skip was designed to be wedged against the lowest set of timber to take the place of the guides below the timber. It was 25 ft. long, and offset 10 in. so as to bring the wheels of the skip back under the timber, close to one side of the shaft, thus allowing space enough to get past the skip on the other side when it was on the bottom

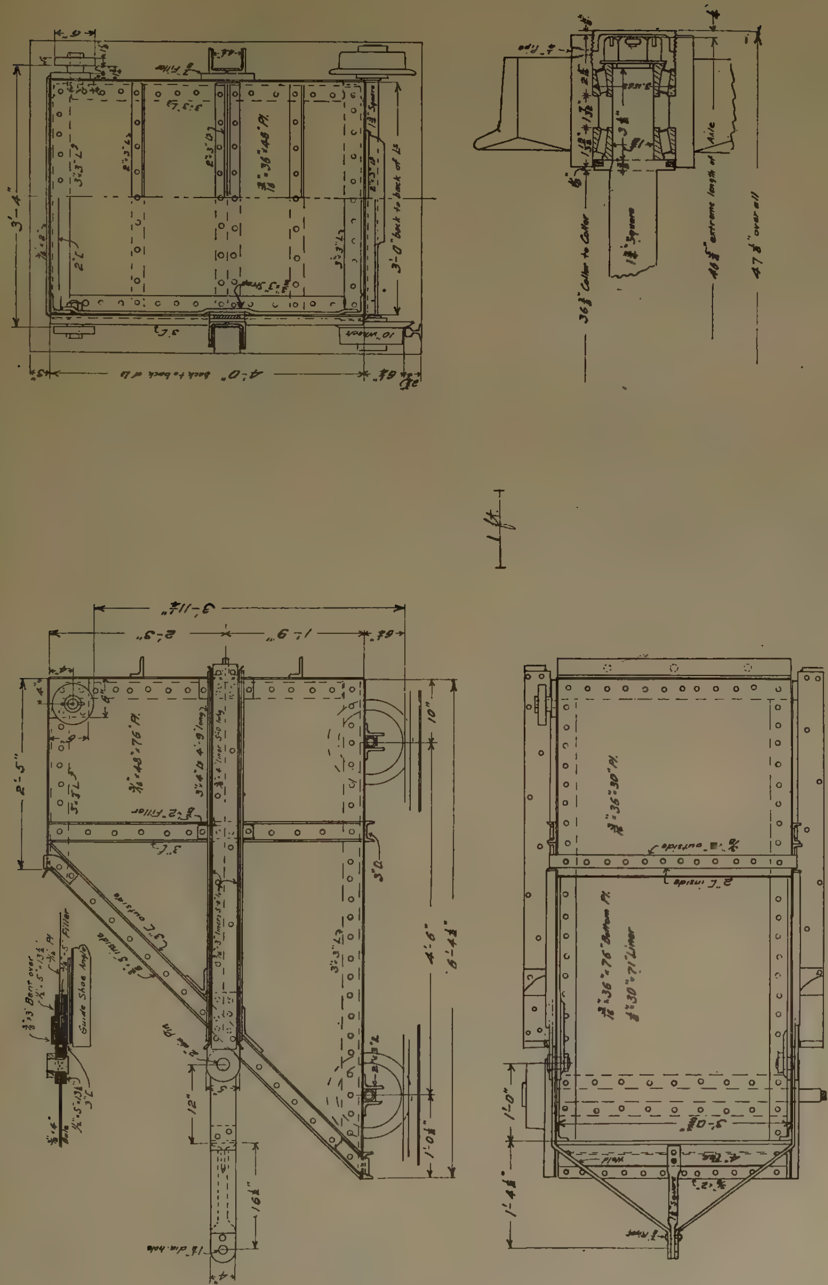

FIG. 2.—SKIP FOR SINKING VERTICAL SHAFT.

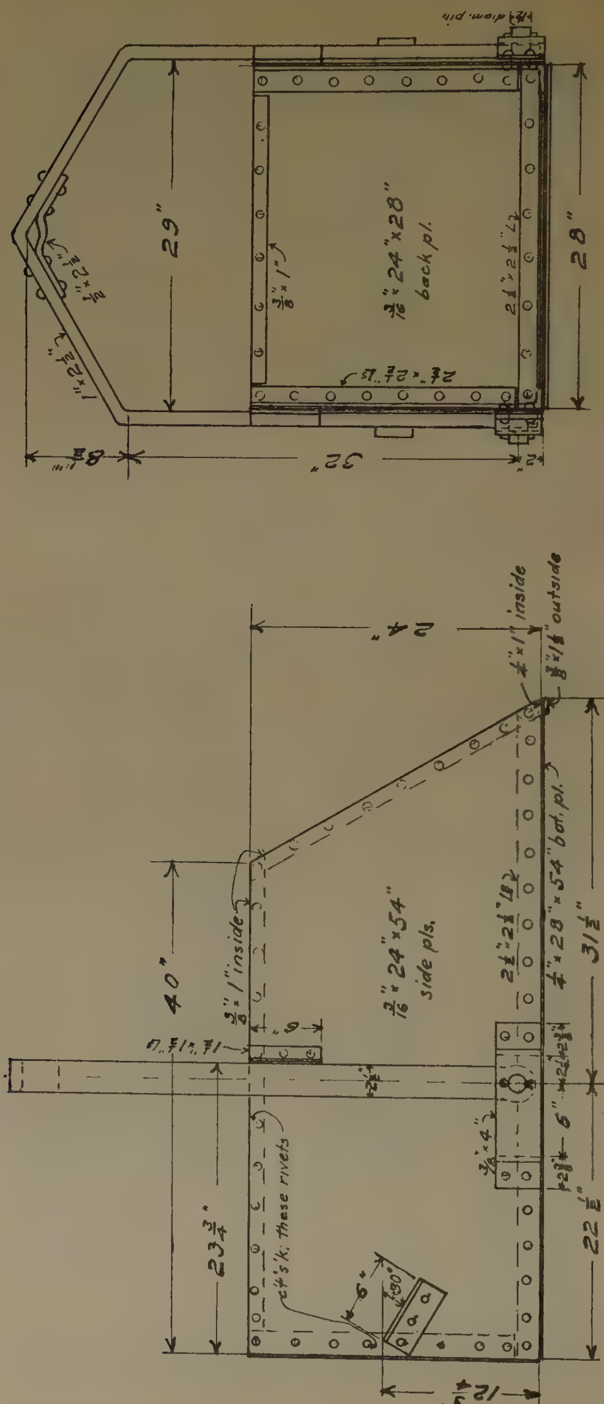

FIG. 3.—LOADING PANS FOR SINKING VERTICAL SHAFT.
From drawing by Timber Butte Milling Co.

(see Figs. 2, 3, 4). Two loading pans were used with a capacity of nearly 1 ton each. These were handled by two 2-ton air hoists hung above the last set of timber, the ropes coming down each side of the center compartment. The hoists were operated from the bottom by two hand ropes on each hoist, one for lowering and one for hoisting. It was found that a single lift rope was necessary, so a hoist that would lift 1½ tons with one rope was required. With both pans loaded while the skip was making the trip up and back, the skip could be filled in less than one minute.

A bulkhead, or blasting set, was first built of 8 by 10-in. timbers, covered with strap iron, but was soon smashed up. Then one was built of 8-in. I-beams with timber and two 30-lb. rails clamped to the lower side. The two end compartments were permanently bulkheaded by timbers laid across this set. The center compartment was closed before blasting by a sheet-iron door opening downward. This bulkhead was hung on a 2-ton chain hoist at each end. Two of these bulkheads, with occasional repairs, lasted for the whole 2000 ft. sunk. It was found that the modern high-carbon steel rails would not answer for protection, but some old soft 30-lb. rail lasted very well.

To avoid delays due to the electric hoist, it was soon found necessary to have a spare rotor

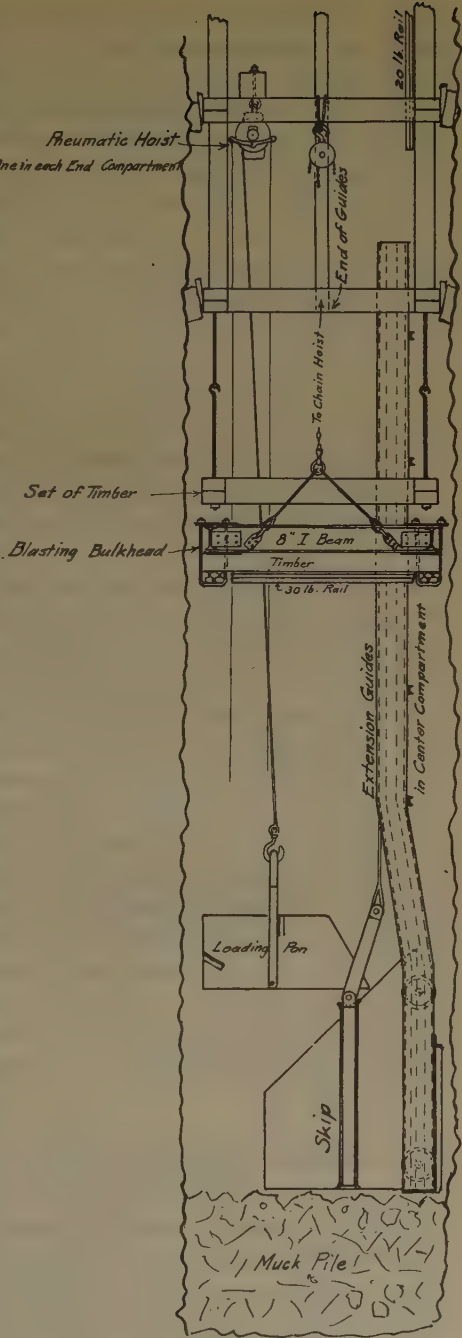

FIG. 4.—RIG FOR MUCKING VERTICAL SHAFT.

on hand. This precaution saved a serious delay when one of the hoistmen lowered an empty skip too fast, with the hoist motor clutched into gear. It was contrary to orders to lower in this way with the current turned off. The result was that the centrifugal force burst the binding wires and threw all the coils out of the slots in the rotor. Although the stator windings were badly damaged and deformed, when the spare rotor was installed the motor functioned, with occasional patching, until the shaft was down. It had to be completely rewound before the cutting of the 8600-ft. station was completed. The hoistmen were paid by the contractors and shared in the bonus for speed. His desire to get the skip back quickly probably caused the man to disobey orders.

Ventilation, Pumping and Compressed Air Line

Ventilation was provided by means of a 12-in. galvanized iron pipe and a reversible positive pressure blower driven by a 10-hp. motor.

In order to be prepared for pumping in case of a sudden flow of water, the permanent column pipe was installed as the shaft was sunk. This consists of 8-in. pipe, flanged and painted and wrapped with painted cloth, starting with casing at the top, then standard, then extra heavy, ending at 1750 ft. From that point down, it is 6-in. extra heavy. The flanges on the extra heavy are 12-bolt male and female with a vulcanized fiber ring gasket.

A 6-in. screw joint casing pipe, wrapped with painted cloth, was permanently installed for compressed air as the work progressed. A 50-ft. length of 2-in hose was attached to the last length, ending in a manifold for the eight 25-ft. drill hoses. The installation of the pipe was easy with the cage in the same compartment as the pipe. For installing the screw joint casing, a taper plug was made to fit the pipe, free to turn on a pivot. When the cage had been lowered to the proper place, this pivot was placed in line with the last length of pipe, the new length resting on top of it. The cage was then raised until the upper end entered the sleeve on the pipe above with some pressure on the threads; when the pipe was turned on the pivot the threads engaged and the joint was easily screwed up tight with chain tongs.

Routine of Sinking

The regular routine of sinking was in three shifts, as follows:

Morning Shift.—The shift consisted of 8 drillmen, one of them boss, I mechanic, 1 tool nipper, and 1 hoistman. The drillers took their machines and hose down with them. Waugh 37 unmounted drills, using 1-in. hexagon collared steel were used throughout. About 40 holes were drilled for an advance of 5 ft. This shift drilled and blasted the round, usually in time for the smoke to be out when the afternoon shift came on. Delay-action exploders were used, fired by the 440-volt power

circuit connected through a circuit breaker set at 50 amp. Current from the 250-volt direct-current supply was found to be unreliable, probably due to leakage of current down the long stretch of wet shaft. The firing switch was located at 40-ft. station until after cutting a pump station 1000 ft. down. After the holes were loaded and connected, the extension guide rail was hung to the bottom of the skip in the center compartment and hoisted above the bulkhead. The door in the bulkhead was closed and the men rode up in the other skip.

Afternoon Shift.—On this shift there were 5 muckers, one of them a lead man, 2 timbermen and 1 hoistman. First, the trap door in the bulkhead was opened and the extension guide lowered. Then the blasting set was lowered to allow room for the next set of timbers. The timbers were then brought down and the wall plates put into approximate position. The extension guide was then wedged into place, after which mucking out began, the timbermen working at the same time on top of the blasting set. After the set of timber was placed and blocked, the blasting set was pulled up against it and wedged tight. About 25 skips of rock were mucked on this shift.

Night Shift.—On this shift there were 5 muckers and 1 hoistman. The mucking out was completed on this shift, consisting of about 25 skips of rock.

Advantages and Disadvantages of System

This system of always doing the drilling and blasting on day shift had the advantage that the men on each shift only had to learn how to do one job. It was not necessary that all the men should be good drillers. The disadvantage was that if a delay occurred on any shift, it meant a loss of 24 hr. If, for instance, the day shift was unable to finish drilling the round, the time lost could never be made up by the other shifts, as the men were less efficient at that work, and the only way to get back to the routine was to put the men at other work and let the day-shift men finish their own job.

Attempts to break more than 5 ft. per round by using more holes and 60 or 80 per cent. powder were not successful. The blasting set was broken several times without greater footage having been gained.

The water was handled by bailing into the skip until the shaft was down 1650 ft., but the shaft was very wet to work in.

Progress and Costs

With no serious delays, this routine resulted in an average advance of 125 ft. per month, with a maximum of 137. The average cost under these conditions was $80 per foot, plus a charge for "overhead" of $20 per foot. In August, 1926, a flow of water and other troubles, which will be referred to later, resulted in an advance of only 35 ft. The cost

under these conditions was $92.50 per foot, plus $107.50 for overhead, total charge $200 per foot.

The contract price for this shaft was $65 per foot for all labor and explosives. The company paid for all other supplies and equipment, sharpening steel and accident insurance, and guaranteed the contractors against loss over any period of one month due to causes beyond the control of the contractor.

The contractor did a good job and everything went well for 14 months. With only about 400 ft. left to go, some soft rock was reached which carried a flow of water amounting to about 10 gal. per min. This happened when the contractor was absent and the foreman left in charge was sick. The men got discouraged and the crew became disorganized. This caused a delay of six weeks or more, and the No. 1 shaft reached the objective first, contrary to expectations.

In order to take care of an influx of water of this kind, a sump and pump station had been cut 1000 ft. down and a triplex pump, capacity 50 gal. per min., geared to a 25-hp. motor, had been installed. The permanent power, telephone and signal cables were brought down to this station. Another triplex pump with a capacity of 20 gal. per minute against a 1000-ft. head was kept ready to be mounted on a 500-gal. tank with guides fitted for the third compartment of the shaft. This was as large a pump and tank as could be gotten into the shaft; it was designed to be lowered to within 100 ft. of the bottom, and to pump out the water delivered to it by a sinking pump. When the water came too fast for bailing, it became necessary to use this rig. It proved unsatisfactory, for the following reasons: The 500-gal. sump was so small that the accumulation of water during the blasting period could not be pumped out quickly enough; also, it quickly filled with sand from the sinking pump. With a steady rain of water coming down the shaft and splashing horizontally from the timbers, the motor and wiring had to be completely enclosed, to keep them dry. In the confined space of the shaft, the cover had to be removed to get at the pump or connections, and then everything got saturated, resulting in leakage of current. Therefore, another pump station and sump had to be cut for this pump. It required 20 days to cut the station, during which time an advance of only 30 ft. was made.

Before connecting with the shaft above, it was necessary to cut the station and bins at the bottom and have everything ready so that the ore could be lowered down No. 1 shaft and hoisted up the vertical shaft. This work, as well as taking out the pillar of ground left between the new and old shafts, was also done by Mr. Johnson on contract.

It required 3 days to connect the timber, guides and rails of the new and old shafts, and to get the longer ropes on the dolly cage and counterweight. During this period, no hoisting could be done. In the main

hoist, 400 ft. less rope was required to reach the 8600-ft. level than to reach the 6300-ft. level.

The shaft, including the preparation and the final connection, cost $230,236, or $118.38 per foot, including overhead. The 8600 station, including pump station, bins and the cross-cut and raise connecting it with No. 1 shaft, cost $33,000.

No. 1 Incline Shaft

By the time the development program mentioned had been finally adopted, No.1 shaft was down about 200 ft. below the 6600-ft. level, with no station on that level, because nearly all the ore had already been taken out through a winze. There was a 50-hp. double-drum electric hoist already installed. This hoist was too small, but it would have been a slow and expensive job to get a larger one into place, so no change was made. Later on, it was fitted with new drums, brakes and clutches.

Ventilation was provided by a 5-hp. electric blower and 8-in. galvanized iron pipe. The exhaust from the sinking pumps actually furnished most of the ventilation.

The No. 1 vein is a fault vein, entirely different from any of the other veins in the mine. It consists of 30 ft. or more of crushed material without well-defined walls, the quartz occurring without much regularity in the vein filling. In some places, there are actual open spaces, which are full of water and drain off fast when sinking. After the water has drained off, it leaves the ground quite dry.

The contract for sinking this and No. 2 shaft was given to Robert Bedford, who was mining engineer and superintendent for the company for a number of years. His partner, James Wood, acted as underground foreman for him. A tentative price of $35 per foot for labor in the shaft was agreed upon, with the understanding that it would be adjusted up or down later on. This price was to include the labor cost of cutting stations every 300 ft. A month later, a new contract set the price at $37.50 per foot for 100 ft. per month or less, and $2.50 per foot bonus for each 10 ft. of advance over 100 feet.

The shaft has two compartments 4½ ft. wide and one 3 ft. wide, both by 5 ft. 4 in. high. Cap and end posts are 10 by 10 in.; center posts 8 by 10 in., and mud sill 6 by 10 in. Braces 6 by 8 in. and studdles 4 by 6 in. Lagging was required almost throughout the shaft. This makes 15 ft. by 6 ft. 8 in. outside of the timber. The contractors placed the timber and laid the tracks and the 6-in. line for compressed air.

The ground in this shaft was soft and had to be timbered close to the face. Two drills were used, mounted on columns. A round was drilled and blasted in a shift, using caps and fuses. It then required three shifts to muck out and timber up. For the first 9 months it averaged 5⅓ ft. of advance per round. This was increased to 6½ ft. when

the ground would stand better without timber. Holes per round averaged 22.

The intention at first was to have two or three men cutting the stations while the shaft was being sunk. This proved to be very much too slow, partly because these men were frequently taken to help out in shaft sinking. The result was that the bottom of the shaft went over 600 ft. below the last station, which made all kinds of trouble. Too long a trip for the hoist delayed mucking, but the worst difficulty was with the pumping. With too high a head for the sinking pumps to work against, the water had to be relayed from one pump to another, as the shaft was too small for installing an electric pump. At one time, there were three sinking pumps, one above the other. As it was almost impossible to keep the three going at the right speed, they were running in fork and pounding themselves to pieces, besides using more compressed air than could be supplied. It finally became necessary to stop sinking until the stations could be completed and a hoist for sinking installed on the lowest station, as had been planned originally. This made it possible for the main hoist to operate in balance, and left it free, part of the time, to hoist ore, which was much needed to keep the mill running.

Under the contract in force during this time, the contractor was paid only for advance made in the shaft, so when it became necessary to stop sinking and cut stations, there was no money coming in. A new contract had to be drawn up, according to which 115 cu. ft. of station cut was considered equal to 1 ft. of advance in the shaft, and paid for accordingly.

After this, everything went smoothly for several months. No sudden flow of water had been encountered and the water that was coming, about 150 gal. per min., was being handled easily by one of two station pumps on the 6300-ft. station. It was desired to replace the smaller of these two pumps with a new and larger one, but to install the new pump before removing the old one necessitated the cutting of a new pump station at considerable expense; so it was decided to take a chance.

When everything was ready, the old pump was removed and the work of building the foundation for the new pump started. The next day the water in the shaft began to increase until it was more than the one station pump remaining could handle. It was two weeks before this difficulty was straightened out and sinking resumed in the shaft.

A little over 18 months was spent in sinking the shaft 1843 ft. to the 8600-ft. station. Two sinking pumps were always kept in the bottom, one running and one ready. The pumps had to be sent to the shop for repairs about every fortnight. When operated against a head of over 100 ft., they always gave trouble.

As each station was cut, an electric plunger pump was installed, with an air pump as a stand-by. An electric pump was left permanently

installed on the 6900 and the 7850-ft. stations, each pumping to the 6300 through a 6-in. column.

No. 2 Incline Shaft

The contract for sinking No. 2 shaft was included in the one with Bedford and Wood for No. 1 shaft. At the time the contract went into effect, the shaft had already been sunk about 600 ft. below the 6000-ft. level, the 6600-ft. station had been cut, and ore was being extracted from that level.

A double-drum air-driven geared hoist on the 6000-ft. level hoisted the rock by means of skips of 30 cu. ft. capacity, to bins on that level. The rock was then trammed 2800 ft. by storage-battery locomotive or mules to the North Star shaft. Ventilation was furnished through a 10-in. galvanized iron pipe by a reversible positive pressure blower driven by a 10-hp. motor.

The No. 2 vein is small with hard walls, like most of the others in the mine, and only occasionally required timber. The minimum size was 15 ft. wide by 7 ft. high. The contractors laid two 30-in. gage tracks, placed what timber was necessary, and installed the necessary pipe.

Because the extraction of ore on 6600-ft. level made it impractical to install a sinking hoist on that station, it was planned to sink with the main hoist until it could be installed on the 6900-ft. station. However, the same difficulty in cutting stations that developed in No. 1 shaft gave as much or more trouble in sinking No. 2. The contractor never completed 69 station, and the 72 station was the first one cut. In order to maintain production, it was necessary to hoist ore from 6600 level at the same time that one rope was being used for mucking out the shaft. This was done by using two hoistmen at the same time. While one man with the brake was lowering a skip to the bottom and waiting for it to be loaded, the other one could hoist a load or two from 6600 with the engine on the other rope.

Very little water was encountered in this shaft—less than 20 gal. per min. at any time. Nevertheless, this small amount gave much trouble when it had to be relayed from one air pump to another before the 72 station was cut. After a sump was built on this station, a triplex pump with a capacity of 30 gal. per min. against a 334-ft. head and driven by a 5-hp. motor, was installed. This pump was started or stopped by a float on the water in the sump and required little attention.

Advance averaged 108 ft. per month, with a maximum of 130 ft. Each round required about 32 holes to break 6½ ft., with one shift to drill and three to muck out.

The cost averaged $83.73 per foot, including an overhead charge of $24.39. The 72 and 75 stations cost about $3000 each.

Connecting No. 2 Shaft with 8600-ft. Level

After the shaft had been sunk 1071 ft. under this contract and there was still 350 ft. to go, it was decided to stop sinking and connect by raising from the 8600-ft. level after it was driven in. This would delay the final connection and the completion of the program about 3 months, but raising would cost much less than sinking, and it was becoming imperative to increase the stoping and development of the No. 2 vein.

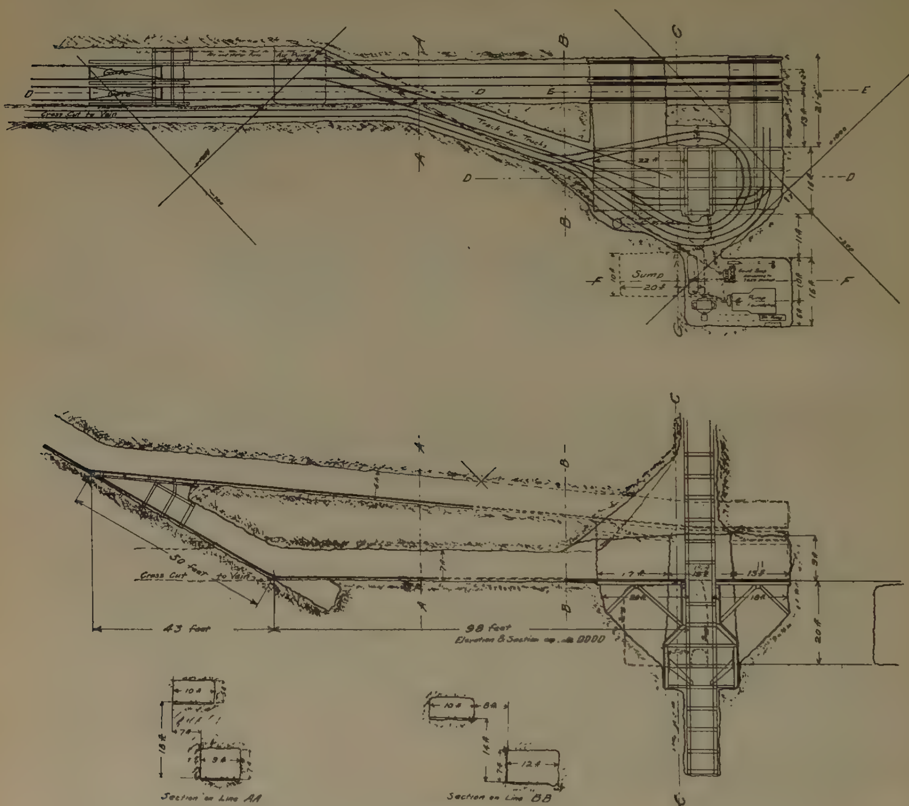

Fig. 5A.

This could not be done while the shaft was being sunk, not so much because of the handling of the rock as on account of difficulty in getting the men and supplies in to the working faces. Everything had to be handled by three different hoists and a 2800-ft. tram. It required 40 min. to reach the bottom of the shaft from the surface when there was no waiting for trucks or cages, and it sometimes took 2 hr. for a mechanic or electrician to do this in the middle of a shift when cages and trucks were in use for moving timber, and so forth.

As soon as possible after the connection from the vertical shaft to No. 1 shaft had been made, the 8600-ft. drift was started to connect

with No. 2 shaft. The 1743 ft. was driven in a little over 6 months. The greatest distance made in one month was 362 ft. The cost averaged $17.85 per foot, of which $6.55 was the charge for overhead.

The raise from this drift to connect with the No. 2 shaft was started on a 45° pitch from a point under the vein so that the lower 200 ft. would serve as a chute. This portion was divided by partitions into a chute for waste, one for quartz, and a manway with a track for the truck.

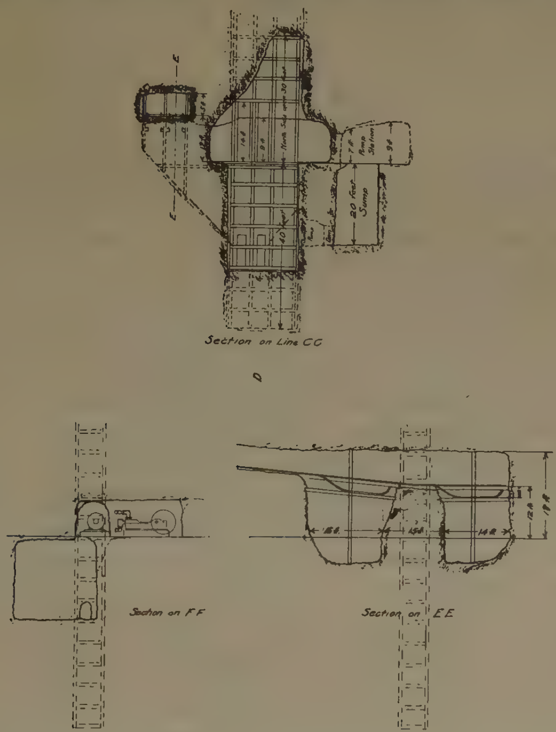

FIG. 5B.

FIG. 5.—JUNCTION OF NO. 1 SHAFT AND VERTICAL SHAFT, 8600-FT. STATION.

It required two months to make the connection and cost $39 per foot, of which $10.50 was the charge for overhead.

During the period of 8 months while the connection from No. 2 shaft to the 86 station of the vertical shaft was being made, all the ore mined from the No. 2 vein, amounting to about 300 tons per day, had to be lowered from the 6000-ft. level down No. 1 shaft. The ore from No. 1 vein had to be lowered down the same shaft; but as this originated around the 7200-ft. level, it was lowered only about half as far. This meant fast work for the small hoist. It was necessary to keep a

stream of water running on the brakes, and the skips jumped the track rather often. Nearly all the track in the shaft had to be laid over again during this time.

Fig. 5 shows the 8600-ft. station. This was cut in the hanging wall of No. 1 vein, to avoid heavy ground. It was planned with some care, to make use of every cubic foot of excavation, at the same time making it convenient for receiving ore and waste from the skips of No. 1 shaft, and the ore train from 8600-ft. level, as well as for the transfer of men and supplies.

Pumping System

Fig. 6 shows the permanent pumping system of the mine. Except in case of emergency, all the water from the 8600-ft. level is pumped in one lift of 1920 ft. through the water column in the vertical shaft to the 4000 station. In case of a sudden flow of water, or trouble with the pump or column, it can be pumped up by stages through the incline shafts. The air pumps are provided in case the electric power fails. On this station, there is storage for the ordinary flow of water for only about 3 hr. The pumps on 7850, 6900 and 6300 ordinarily are run for a short time once a day to catch up the water collected in the dams, so that it will not all have to be pumped from the bottom.

The electric power is supplied by duplicate lead-covered cables, one going straight down the vertical shaft to the 8600-ft. level, the other by way of the vertical as far as 4000 station, then through the two incline shafts. These cables ordinarily run in parallel, being connected at the top, the 4000 and the 8600; but either one can carry the full load alone.

Drifting during Shaft Sinking

When the shaft sinking started, there was only enough ore developed to last three or four months; so, as the shafts were sunk, drifts had to be driven. The first year about 5000 ft. were driven; the second, 12,000, exclusive of the shafts. During the two-year period, 242,900 tons of ore was mined, producing $1,518,000.

Total cost of the 3 shafts	$559,106.00
Cost of other development during 26-month period	322,975.00
Total of all development	$882,081.00
Total deficit for 26 months	708,531.00
Balance	$173,550.00

The extraction of ore not only lessened the total net outlay by the amount of $173,550, but also by paying a portion of the fixed charges, which would amount to about one hundred thousand dollars more.

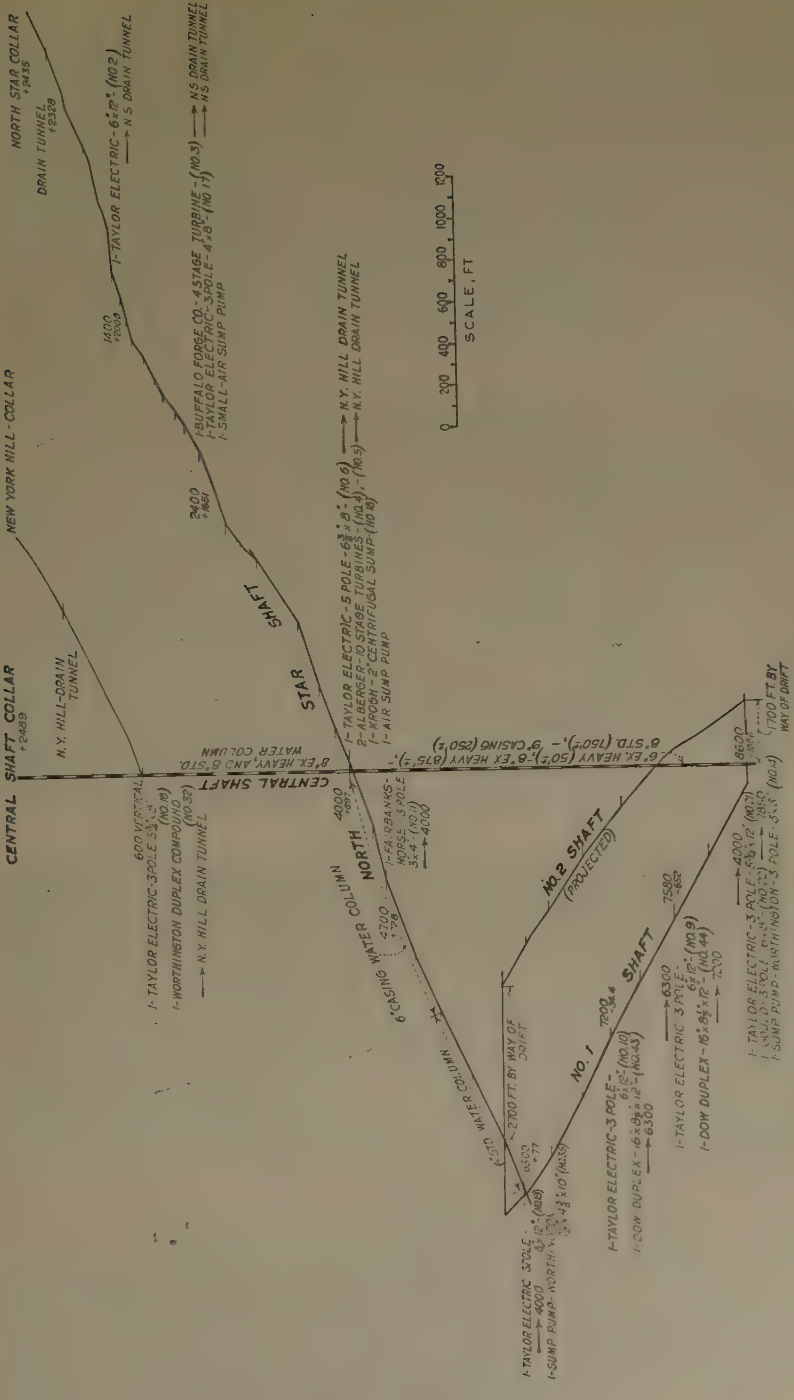

FIG. 6.—PUMP LAYOUT.

Conclusions

In looking back over these operations, the use of the blasting set in sinking the vertical shaft seems worthy of comment, as it saved 2 or 3 hr. time each round. First, it enabled the timbermen to do their work at the same time that mucking was going on. Second, being a tight bulkhead below the lowest set, there was no time spent in cleaning the loose rock off the timbers after each blast. With men working at the bottom of a shaft of this depth, it was particularly important that there should be no rock to be jarred loose by hoisting. It was also necessary to have a skip that would not spill rock either in the vertical or the incline shaft. No accident from falling rock occurred, and no fatal accident in any of the shafts.

Two delays that might have been avoided occurred in sinking the vertical shaft, and one very serious problem caused expense and delay in sinking the incline shafts.

The accident to the hoist caused by overspeedng suggests that there should have been a clause in the contract making the contractor bear the expense of a breakdown caused by the neglect of his employees.

The difficulty of pumping water from sinking operations against a high head is not easily overcome. In the light of the experience gained it might have been solved without cutting a pump station, if the pump and tank combination had been more carefully designed. It is possible that a multistage centrifugal pump constructed to handle gritty water and driven by a waterproof motor could have been obtained. The space was limited and the weight with the tank empty had to be within the capacity of the hoist.

The failure to cut the stations in the incline shafts in time, so that the bottom of the shaft got too far below, might have been avoided by a properly worded clause in the contract setting a definite limit to the distance below the lowest station for the shaft bottom.

With hoisting going on in four shafts at top speed, it was inevitable that wrecks and breakdowns should occur, necessitating long hours of overtime for the repair men and bosses; and too much credit cannot be given for the way they did their work. The track in the main hoisting shaft was none too good, and derailments caused much damage, especially when a skip off the track was pulled around the turn into the vertical shaft.

Acknowledgments

R. W. Parsons, the superintendent, and his assistant, who did the surveying, J. R. C. Mann, especially deserve credit. It was hard to keep things running smoothly when the shafts were given preference over everything, and the shaft men were paid higher wages than the company bosses. There was not much time to spend on surveying, especially in

the shafts, but the vertical and No. 1 shaft connected within 6 in.; and the connection with No. 2 shaft, which involved a traverse of 4400 ft. of crooked drift, plus 4300 ft. of shaft, closed within 3 inches.

J. Fred Johnson organized the crew and developed the methods for sinking the vertical shaft, and deserves especial credit for the good alignment of the shaft; also for his direction of the critical job of connecting with the shaft above. To R. H. Bedford belongs the credit for organizing and directing the sinking of both incline shafts.

Some Recent Developments in Open-pit Mining on the Mesabi Range

By Earl E. Hunner,* Duluth, Minn.

(New York Meeting, February, 1930)

It is common knowledge that the iron orebodies of the Mesabi Range lie nearly horizontal and are of trough or blanketlike types. These orebodies are from a few feet to several hundred feet thick and vary from a few acres in extent to the size of the largest one, at Hibbing, Minn., which is over 1 mile wide and 3 miles long. They are covered with sand, boulder and clay overburden and occasionally shallow depths of ledge material, varying in total depth from about 20 to 200 ft. Large areas are not capped with overlying ledge material; the top of the ore lies directly underneath the surface material. Because of the comparatively great tonnage of ore, open-pit mining has made tremendous strides since the first open-pit ore was shipped from this range by the Merritt brothers, from the Mountain iron mine in October, 1892, and along with the great development of open-pit mining on the Mesabi Range has gone a tremendous change in the character of equipment used.

Early Stripping Operations

The first stripping at the Burt mine at Hibbing, Minn., started in 1902. The engineers spent a long time in selecting the most suitable location for a small pit area, involving a stripping contract of 350,000 cu. yd. This property afterwards had several million yards of stripping and several million tons of ore removed from it, but inasmuch as it was the first open pit of the Oliver Iron Mining Co. in this district it was necessary to try several different locations for the small pit area, to uncover maximum tonnage and the best grade of ore so as to sell the idea of open-pit mining and discontinue the underground operation then going on.

The writer's first open-pit work on the Mesabi Range was in 1903 as an Oliver company mining engineer on a stripping job of 1,500,000 cu. yd. at the Monroe mine, Chisholm, Minn. Here the contractor, an old-timer used to railroad methods, was, to a small extent, using teams of horses hitched to 2-yd. narrow-gage cars and hauling the stripping material from the pit area to outside of the pit limits. The dirt thus moved was loaded by hand. The same contractor was also using 60-ton and 70-ton railroad-type shovels with dippers of 2½ and

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3 cu. yd. capacity, loading material into wooden narrow-gage cars of 4 cu. yd. capacity, hauled by saddle-back steam locomotives of 8 to 18 tons. This type of power equipment was in common use at that time on stripping operations. Shovels of the same type were used for loading ore into the railroad cars, switched from the storage yard located on the natural surface of the ground not far from the pit, down through the approach to the pit, to the top of the orebody, by steam locomotives of 45 to 70 tons, of the six-wheel switcher type having up to 11 ft. wheel base. As mining progressed and the ore pit was deepened, additional tracks were laid downgrade, either as switchbacks on one side of the pit or spiraling around the pit, so as to follow the surface of ore, as it was mined and lowered, and enable the loading and hauling of ore from the pit to continue.

Some of the pits started about that time and some that have been started more recently are now attaining considerable depths, and mining methods as well as equipment are considerably changed to meet the more exacting demands of greater production requirements, and greater depths at which mining is conducted today.

Steam Shovels Replaced by Electric

In 1914, when the writer was in charge of operations of the Great Northern Iron Ore Properties, the largest stripping contract ever made on the Mesabi Range was let to open the Hill Annex mine at Calumet, Minn.; amounting to 8,000,000 cu. yd. of stripping, at the rate of 2,000,000 cu. yd. annually, to make available for mining 6,000,000 tons of ore. The plan was to follow up this program later with additional work until a total of about 22,000,000 cu. yd. of stripping would be moved and 25,000,000 tons of ore mined, with possibilities of even more material being handled, depending on costs and market conditions. When the work was in full swing in 1917, the contractors were using the following equipment, which compared favorably with equipment on other jobs of the same time:

9 six-wheel Baldwin switching steam locomotives 20 by 26 in.
8 Kilbourne and Jacobs 20-yd. automatic air-dump cars of 20 cu. yd. capacity.
49 Kilbourne and Jacobs 16-yd. automatic air-dump cars of 20 cu. yd. capacity.
1 eight-wheel, 25-ton Industrial Work locomotive crane boom, radius 50 ft.

All these were equipped with Westinghouse air brakes.

1 model 100 Marion steam shovel, 5 cu. yd. dipper, 137-ton weight.
2 model 76 Marion steam shovel, 4 cu. yd. dipper, 107-ton weight.
1 model 36 Marion revolving shovel, 1½ cu. yd. dipper, 40-ton weight.

By 1919, this job had added one 300-ton revolving steam shovel, using 6 and 8 cu. yd. dipper. One of these shovels made a double-shift, one-month record of nearly 250,000 cu. yd. of sand and gravel.

The tracks were laid of 80-lb. steel. Track grades used in moving the stripping out of the pit were from 1.5 to 2.5 per cent. and out on to the stripping dumps were 1 to 2 per cent. The main-line ore track out of the pit was 2 per cent. The surface overburden on this job averaged well over 100 ft. deep, being 150 ft. in the deepest part of the 8,000,000 cu. yd. pit area, which required excessive grades on a limited yardage to clean out a deep wash channel.

Prior to 1919, there were only about six 300-ton revolving steam shovels in use. Our experience, prior to 1919, at the Wakefield mine, Wakefield, Mich., operated by The M. A. Hanna Co., had proved to us that the use of this large type of equipment had decided advantages under certain conditions over the 100 to 110-ton railroad-type shovel at that time still in general use for heavy work. Since the earlier purchases of this type of large revolving shovel, a number have been put to work on the Mesabi Range.

Our use of electric power in many different instances, such as underground haulage, hoisting, pumping, air compressors, etc., as compared to steam power, made it imperative to consider seriously the power question, and the effect of its use on labor, repairs and other costs, and as a result of considerable study, our company purchased in 1919 one Marion 300-ton E-type, and one Bucyrus 225-B revolving electric shovel, each equipped with 150-ft. boom and complete dragline equipment. Both shovels had 8 cu. yd. buckets for use in stripping and 6 cu. yd. buckets for use in ore. The draglines each had Page buckets of 5 cu. yd. capacity. Neither manufacturer could manufacture both shovels in time, so the advantage of having two similar pieces of equipment was sacrified by splitting the purchase order.

The power supply furnished by the power company is three-phase, 60-cycle, 22,000 volts. A bank of three single-phase transformers steps this current down to 2200 volts; this is carried by pole line into the pit and conveyed to the shovels by means of flexible cables.

This equipment has been in service 10 years and is in excellent shape today. A noticeable, favorable comparison to the same type of steam shovels is the repair cost. The ratio is about one to four, in cases where it has been possible to get the steam-power experience of other operators.

With one of the shovels equipped as a dragline, we stripped the surface overburden from the Shiras pit, at Buhl, Minn., and loaded out all the ore, exhausting the pit. In this pit the surface material averaged about 25 to 30 ft. deep and the underlying ore about 40 ft. The ore exposed after stripping was 125 to 200 ft. wide and 1500 ft. long, and was exceedingly uneven on both the top and bottom of the deposit, so much so that if mined by ordinary shovel and locomotive haul methods, it would have been impossible to avoid mixing of surface and ore without consequent loss of ore, and neither could the bottom of ore be all mined

clean without much greater expense. Underground mining costs would have been prohibitive, because the long, narrow deposit would not have permitted the use of enough miners to offset overhead charges. The dragline was set up on top of the ground near one end of the proposed pit area and moved along the south side, overcasting surface from about two-thirds the width of the pit. Then it advanced around the end of the pit to the north side, and removed by overcasting the remaining third of stripping. Grade and alignment stakes had been placed for a loading track paralleling the north side of the pit on 1.25 per cent. grade. This track grade was built as the dragline progressed along the pit edge, and surplus dirt was overcast beyond the railroad fill. When stripping was completed, the dragline was back at its original location, at the end of the pit. It then cut its way down on to the top of the orebody and progressed to the far end of the pit, on top of the ore, where it tied up for winter. The following spring it started draglining ore, loading direct, with lift of 80 ft. from bottom of ore deposit to top of car, into standard 50-ton and 75-ton cars, on loading track above it, with practically no spill. The operator slowly lowered the bucket into the car body to get the correct position, and then lifted vertically to clear the bucket before tripping. Rock horses or intrusions were encountered frequently in the ore at times and this rock was separated by dumping the loaded bucket at the top of the ore slope and directly in front of the operator. The coarse rock in falling would bounce down the ore slope where it would be picked up by rock pickers and piled at one side on the rock bottom that had been cleaned of ore. In places the ore in the bottom of the trough slowly graded into soft bottom rock, too lean to mine. The fee owners' inspectors occasionally wanted proof that no ore existed lower down, and this was supplied by trenching deeply with the drag bucket. When not necessary to have rock pickers, the crew consisted of one operator, one oiler, one ground man and two car men whose duties were to trim cars when loaded, release car brakes and ride cars when dropping by gravity to location in storage yards, where they were stopped by hand brakes. The average production per 10-hr. day was about 1250 tons.

Since the purchase of these two large electric shovels in 1919, three other similar electric shovels have been installed in the Lake Superior district and smaller electric shovels have also been gradually introduced.

In 1924, a Bucyrus 50-B revolving electric, caterpillar traction shovel, with 1¾-yd. bucket, was installed at the Wabigon mine, Buhl, Minn., where the Marion 300-E is working; and in 1925, a Bucyrus 80-B revolving electric, caterpillar traction shovel with 2½-yd. bucket was installed at the Richmond mine, Palmer, Mich. In 1926, two Bucyrus 120-B revolving electric, caterpillar traction shovels with 4-yd. bucket, were installed at the Susquehanna mine, Hibbing, Minn. These three instal-

lations were the first of their type and size, using direct-current motors, in the Lake Superior district.

In 1927, one more of the 120-B types was installed at the Susquehanna mine and in 1928, one of the same type at the New Mesabi Chief mine, Keewatin, Minn., followed by another in 1929. Since the first purchase of the 120-B 4-yd. shovels in 1926 by our company, a number of this size, both Bucyrus and Marion makes, have been purchased and installed on the Mesabi by other companies. In the same time, only one or two steam shovels have been installed, and it is fair, therefore, to state that Mesabi open-pit operators are fully "sold" to the use of electric shovels, with caterpillar traction, preferably revolving type, using 4 to 5 cu. yd. bucket, for general use in heavy service. Over 20 revolving or semirevolving electric shovels have been placed in service on the Mesabi Range during the past year, seven of them using dippers of 5 cu. yd. capacity.

Standard-gage Electric Locomotives in Open-pit Mining

In 1924, the first installation of standard-gage electric locomotives in open-pit mining in this country was made at the Wabigon mine, Buhl, Minn. The equipment installation at this property consists of the following:

Electric Shovels.—One Marion 300-E and one Bucyrus model 50-B. Both machines are equipped with direct-current motors operated from motor-generator sets. In the model 300-E, the control is full Ward Leonard, and in the 50-B it is partly rheostatic. The power supply is carried at 2200 volts, three phase, 60 cycles, on a pole line into the pit, and conveyed to the shovels by means of flexible cables.

Electric Locomotives.—These consist of three 60-ton General Electric swivel-truck type, designed to operate at 600 volts direct current with a capacity to haul a train of 340 tons (including locomotive) up a tangent track of 3 per cent. grade at 7½ miles per hour. Each locomotive is equipped with an air-operated pantagraph sliding-bow collector mounted on the cab roof for use on main tracks, and with two air-operated side-arm collectors mounted on either side for use with side trolley wire on loading and dump tracks.

Trolley-wire System.—After much study and discussion it was decided to use an overhead trolley wire system at 600 volts direct current, which was adopted on account of first cost, freedom from obstruction in the case of derailments, also from obstruction in the case of stripping spill on tracks in the construction of stripping dumps, greater flexibility and general safety as compared to the third-rail system. On main tracks and ore yards, the trolley wire is suspended over the center of the track at a height of 22 ft. above the rail. On temporary

tracks in the ore pit, and also on the stripping and lean-ore dumps, which are continually being shifted, the trolley wire is mounted at the side of the tracks at a distance of 10 ft. from the center of the track and 15 ft. above the top of the rail, the current being taken therefrom by two side-arm collectors, each mounted on either side of the top of the cab, the wire being suspended from portable supports, easy to move about the operation as required.

The power supply in the substation is three-phase, 60-cycle, 22,000 volts. This is purchased from the Minnesota Power and Light Co. A bank of three single-phase transformers steps down the incoming voltage to 2200 volts for the use of the shovels, and six transformers in two banks step down the incoming voltage to 445 volts for the synchronous converters, which supply 600 volts, direct current, to the trolley haulage system.

Formation at Mesabi Chief Mine

Our experience obtained from this installation has been so satisfactory that during 1928–29 we made a similar installation at the Mesabi Chief mine, Keewatin, Minn., which is the second fully electrified open-pit mine on the Mesabi Range. About one-half of the tonnage that will be shipped from this property is direct shipping merchantable ore, and about one-half is wash ore concentrates.

To produce the wash ore concentrates, a new type of screening plant and an improved washing plant have been built, with a capacity up to 380 tons crude per hour, or about 300,000 tons concentrates, single shift, per season. The proposed pit area, roughly rectangular in shape, contains about 75 acres. The surface overburden averages about 30 to 40 ft. deep. Underlying this are four alternating layers of merchantable ore and paint rock, each averaging 10 to 15 ft. thick, and overlying wash ore material about 40 ft. thick. The whole mass of the ore formation dips to the southeast about 8°.

In the northwest corner of the pit area, the surface material, consisting of sand, gravel and clay, directly overlies the wash ore, and as one progresses southward walking on top of the ore formation after the stripping has been removed, one crosses in succession the thin, north feather edges of each of the four ore and paint rock layers which lie above the wash ore layer. This rather unusual stratification in the ore formation made it necessary to start stripping on the north and west sides, the ore cuts following in like manner, so that the shovels work in a northeast-southwest direction and obtain cleanest possible separation between the various layers with minimum degradation, due to mixing of paint rock and good ore, whose value would be destroyed if mixed, and with resulting maximum tonnage recovery.

Pit Equipment at Mesabi Chief Mine

Much study was given to possible benefits from the use of large draglines instead of shovels, with the idea of mining out clean a thorough cut of the crude wash ore in the northwest portion of the property where it directly underlay the surface overburden, and then back-casting the paint rock layers, rather than to haul this material 1½ miles from the pit and stockpile it separately for use if it should ever become of value.

It was finally decided, however, that the best balanced operation—that is, the one that would give the lowest unit costs in this particular operation—would be obtained by using the 120-B caterpillar traction electric shovel and 4 cu. yd. bucket. Much study was given also to the type of electric locomotive to be used. Because dirt stripping, paint rock capping, direct shipping ore, and wash ore to be concentrated, to be moved annually from the pit after the year 1929, would amount to only 1,250,000 to 1,500,000 tons in an eight to nine-month operating season, and the haul to dumps, ore yards and washing plant respectively would average only 2 miles on 1 to 2 per cent. track grades against the load. Furthermore, the cost of electric current is comparatively high, and by adopting comparatively light locomotives and trains, we could keep down peak loads, which was essential. Serious consideration was given to the idea of having manufacturers build a motorized dump car of large capacity to be used as a locomotive, hooked up to ordinary dump cars. To move the direct shipping merchantable ore from the pit to the shipping ore yards, this motorized car would have to be loaded to give it necessary weight and traction to haul standard railway cars loaded with shipping ore out of the pit; this step seemed a little too radical to the pit operators, and finally electric locomotives of regular type were purchased. The suggestion has been successfully worked out in cars of smaller capacity at the Susquehanna operation at Hibbing during the past year, but the next electric locomotive purchased probably will be a motorized dump car.

The pit equipment finally decided on and put to use at the Mesabi Chief consists of the following:

Electric Shovels

One Bucyrus model 225-B, 90-ft. boom, 8 cu. yd. dipper, full revolving type, and two Bucyrus model 120-B, 32-ft. boom, 4 cu. yd. dipper, caterpillar traction, full revolving type. All machines are equipped with direct-current motors operated from motor-generator sets and having Ward-Leonard control.

Power supply is carried at 2200 volts, three-phase, 60 cycles on pole lines in the pit and conveyed to the shovels by means of flexible cables.

During the past two years, the model 225-B shovel has been loading stripping overburden, the train of empty cars standing on the top of

the bank on the natural surface of the ground and the shovel standing on top of the ore formation and loading from this position directly into the train standing above. In this manner, the haul up track grades out of the pit, usually incurred in stripping orebodies, has been eliminated. This shovel is now working as a dragline equipment with 5-yd. bucket and overcasting dirt from within the pit to an overcast pile as far from the pit as the dragline can reach. When this cut is completed, the dragline will go around the pile and overcast again. This second overcasting of the pile will be well without the final pit limits. After the overcasting of the pile is completed, the dragline will dragline again from within the pit and up to the final pit limits, casting this dirt against the first overcast pile. Next fall when this work is completed, this machine will be removed from the property and all work to be done at the property will thereafter be done with the two Bucyrus 120-B machines.

Electric Locomotives

Four 60-ton General Electric, eight-wheel double-swivel, truck-type electric locomotives are used (Fig. 1), equipped with four type HM-

FIG. 1.—60-TON ELECTRIC LOCOMOTIVE WITH SIDE-ARM COLLECTORS AT POINT OF TRANSITION FROM ONE SIDE TO THE OTHER.

This occurs when locomotive is going from permanent to temporary trolley lines, both in the pit and on the stripping dumps.

833 165-hp. 600-volt, direct-current, traction motors and 7° helical gears, giving a continuous tractive effort of 7400 lb. at 14½ miles per hour, 1 hr. rating; 27,200 lb. tractive effort at 8.8 miles per hour. The wheel base is 14 ft. 10 in. as compared with 18 ft. 3 in. on the Wabigon locomotive.

Each locomotive is equipped with type M, two-speed control arranged for 10 points series parallel and 8 points parallel operation of motors. Standard Westinghouse combined automatic and straight air-brake equipment is used, with the air compressor mounted in the cab. All

control contactors and resistors are in a separate compartment inside the cab. A single control station is fitted in each cab, as the experience in the Wabigon proved that double stations are unnecessary and add to the first cost of the locomotive. Standard short-shank locomotive-type couplers have been adopted, as being equally satisfactory and lower in first cost than the long-shank type used formerly.

At the Wabigon mine, standard catenary overhead trolley wire construction was used on the main approach tracks and ore yards with the side-arm trolley system in the pit proper and on stripping dumps. Experience at the Wabigon demonstrated that the side-arm system is satisfactory for all purposes and cheaper to install than the overhead; consequently, at the Mesabi Chief mine, the entire trolley system was laid out for side-arm collection (Fig. 1), using 600 volts, direct current, and the trolley wire placed 10 ft. from center of track and 15 ft. normal height above the rail as formerly. The wire is carried from brackets on poles in all permanent track work and from portable supports where track is temporary, such as in the pit or stripping dumps. Two side-arm collectors are mounted on each locomotive cab, one on each side. These are of an improved design, being constructed in the form of a pantagraph so that the collector shoe always remains in a horizontal position. They are operated by compressed air through valves located in the cab.

Electric Substation

Power for the operation of the electric haulage system is furnished from a substation approximately in the center of gravity of the electric load. This substation maintains one six-phase, 500-kw., 600-volt direct-current synchronous converter with partial automatic switching equipment, including automatic reclosing breakers on the direct-current feeders. There are two banks of transformers, one supplying the synchronous converter and the other a 2200-volt circuit for electric shovels. Three-phase, 60-cycle, 22,000-volt power is supplied to the mine by the Minnesota Power and Light Company.

Dump Cars

The rolling stock comprises 16 air-dump side-pivot drop-door cars of 30 cu. yd. capacity build by the Differential Steel Car Co. (Fig. 2), containing all improvements incorporated in the most up-to-date dump car designed. These cars are used to handle overburden during stripping operations and also haul paint rock from the pit to stockpiles near the washing plant, and crude wash ore from the pit to the washing plant. A Bucyrus heavy-type spreader is used on the dumps and a Nordberg track shifter for throwing tracks and general utility work.

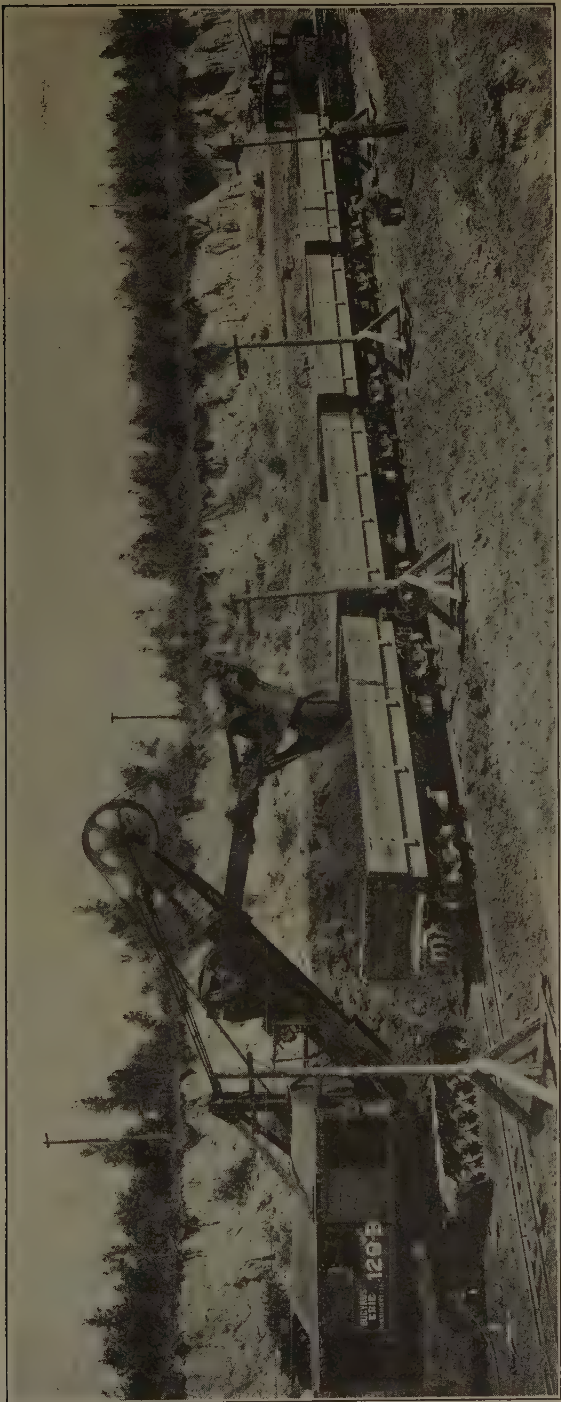

FIG. 2.—BUCYRUS 120-B SHOVEL AND 60-TON ELECTRIC LOCOMOTIVE HAULING TRAIN OF 30 CU. YD. DIFFERENTIAL CARS, ALL EQUIPMENT OF LATEST DESIGN.

Screening Plant

The new screening plant built at the washing plant is of original design, differing from other installations of the same nature on the Mesabi Range. This plant was built by the Link Belt Co. of Chicago. Crude wash material is hauled from the pit to this screening plant in 30-cu. yd. air-dump cars and dumped from the trestle extending over the screening plant on to fixed grizzly bars, inclined at 40° and spaced to give 4½-in. clear openings (Fig. 3). These bars have a screening area of 216 sq. ft. and resemble the sloping sides of two ore pockets. By the use of two spreaders, placed to one side and immediately below track, the entire screened area of the bars is made effective. The material

FIG. 3.—SCREENING AND WASHING PLANT AT MESABI CHIEF MINE.

falling from the car on to the spreaders forcibly collides with material falling directly on to the side and center bars, which results in a rolling as well as a sliding action, in the mass. This tends to break up large chunks and free adhering fines. Oversized material is discharged from the fixed grizzly bars on to two sets of shaking bars (Fig. 4) inclined at an angle of 19°, also spaced to give 4½-in. clear openings. Each set of shakers consists of two units moved in opposite directions by suitable eccentric drives with heavy flywheel, and this opposing action tends to prevent any swaying action in the structure. The upper set of shakers has an area of 61 sq. ft. and the lower 47, making a combined screen area of 216 sq. ft. The lower sets are installed with head ends 2 ft. 6 in. under discharge of upper sets. This drop from the upper to the lower sets in connection with the opposing movement of the bars imparts a rolling action to the chunks, which further tends to free any fines adhering to them. Both the fixed and shaking bars are made of manganese steel

and designed with a special tapering cross-section to eliminate clogging of material. The fixed bars have a depth of 10 in. and the shaking bars 8 in. Undersized material discharging through both fixed and shaking bars drops into a combined pocket of 200 ton capacity, while the oversize discharges into 20-cu. yd. air-dump cars for removal to the waste dump. Three Link Belt, heavy duty-apron feeders, each driven by separate motor, draw the material from the pocket and discharge it on to a link 30-in. horizontal conveyor, which in turn discharges on to a Link Belt 30° inclined belt conveyor running up to the top of the washing plant.

FIG. 4.—SHAKING-SCREEN DISCHARGE.

All troughing and idlers on both conveyor belts are of the Link Belt antifriction type, equipped with Timken roller bearings. The flow sheet is shown in Fig. 5.

Washing Plant

In the washing plant (flow sheet, Fig. 5), the material is discharged from the conveyor belt to an Allis-Chalmers conical revolving screen, 52 by 87 in. by 14 ft. 6 in. long, having 1¼-in. perforations. Oversize from screen passes over a 36-in. picking belt to a 4-ft. Symons cone crusher. Crusher discharge and undersize from screen go to a 16 by 30-ft. Dorr-Davis washer.

The rakes on the Dorr machine discharge directly into the shipping pocket, and oversize from Dorr trommel is carried to pocket by 24-in. Link-Belt belt conveyor.

A Dorr bowl classifier, 22 ft. dia. with 4-ft. rakes, is being installed to treat the overflow water from the Dorr washer, and the fines recovered will be carried to pocket on the 24-in. belt conveyor handling Dorr trommel product.

All motors in the washing and screening plants operate at 440 volts, with the exception of that driving the Symons crusher, which is operated at 2200 volts. The motors are remote-controlled through push-button stations using magnetic control with time limit acceleration and thermal overload protection; the controls are mounted in steel cabinets and circuits run in conduit.

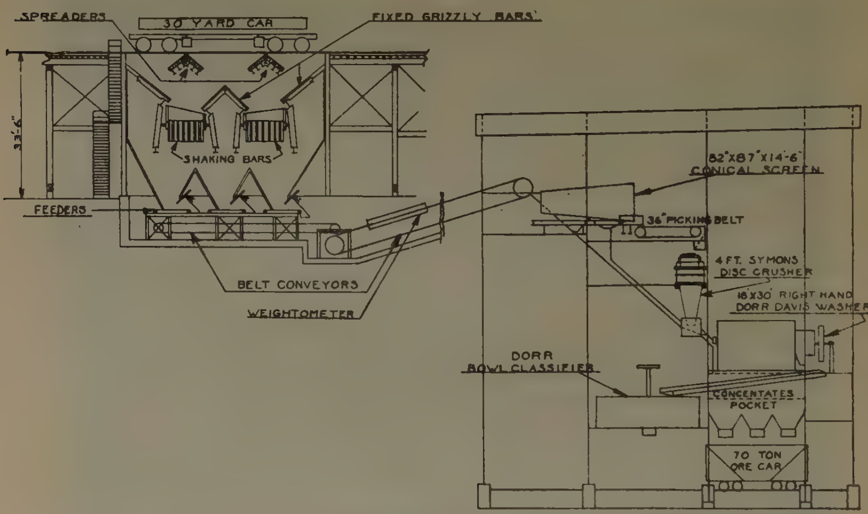

Fig. 5.—Flow sheet of screening and washing plant at Mesabi Chief mine.

The controls are electrically interlocked so that if any motor stops all apparatus ahead of this machine is automatically shut down, thus preventing the feeding of material into the unit that is out of commission. Apparatus behind the machine shut down continues to operate. When starting, the machines at the end must be started first, and the others in progression towards the head of the plant. Herringbone gear, or worm gear reducers are used throughout the plant.

A transformer station is adjacent to the washing plant, which is supplied from a 22,000-volt line running from the main substation at the mine. Thus the plant load is incorporated with the remainder of the mine load, and the diversity of the two is used to advantage.

This installation differs from the ordinary Mesabi Range washing plant in that the Dorr-Davis washer and the Dorr bowl classifier take the place of log washers, turbos and tables. This installation will do more than the old type of installation did, and at less cost, particularly labor, six men being required per shift, as compared to about double

this number per shift in the old type of mill of the same capacity, including the two men required to look after the concentrate loading and dropping by gravity loaded cars from concentrate pockets to storage yards.

Susquehanna Mine

The Susquehanna open-pit mine, near Hibbing, Minn., covers about 75 acres in area. Average depth of surface stripping prior to the opening in 1913 was about 125 ft. deep and of ore within the pit limits, 250 to 300 ft. deep. Prior to Jan. 1, 1926, when The M. A. Hanna Co. of Cleveland took over the operation of the property for the Republic Iron and Steel Co., Inland Steel Co., and its own interest, the total surface stripping removed amounted to 10,360,000 cu. yd., rock stripping 830,000 cu. yd., and all ore shipped 8,590,000 tons. Since then, there has been removed 629,000 cu. yd. surface, 1,360,000 cu. yd. rock and 2,272,000 tons of ore shipped. The bottom of the pit is now down 320 ft.; during the past season, ore was hauled by steam locomotives from the bottom of the pit to ore-storage yard on top of the ground, over approximately 2 miles of 4 per cent. grade. Continuing deeper with the same operation meant steadily mounting costs.

When our company took over the property we largely equipped temporarily with spare equipment from our other mines, as we realized that at the end of three or four years, we would have to change mining methods and equipment also. That equipment consisted of the following:

Used Equipment:

- 1 model 36 Marion revolving caterpillar traction steam shovel.
- 2 model 70 Osgood steam shovels.
- 1 model 85 Bucyrus.
- 5 six-wheel switcher-type 60-ton Lima steam locomotives, 11 ft. wheel base.
- 3 six-wheel switcher-type 60-ton Baldwin locomotives, 11 ft. wheel base.
- 16 Western air-dump cars, 12 cu. yd. capacity.
- 18 Kilbourne and Jacobs air-dump cars, 20 cu. yd. capacity.

New Equipment:

- 5 Differential dump cars, 24 cu. yd. capacity.
- 2 Bucyrus 120-B revolving electric shovels, caterpillar traction.

In an endeavor to produce from 500,000 to 1,000,000 tons of ore annually in a seven months' open shipping season, and to keep future costs at a minimum, not only proper mining and handling of heavy tonnage of open-pit ore remaining were involved but also proper drainage and control of a heavy water problem. The latter required a radical change in the operating plan, which had been put into effect this past season.

The ordinary milling system of sinking shaft, driving haulage drifts out under the pit area, and putting in upraises to surface of ore, through which to mill down the ore to haulage level, could not be adopted here for various reasons.

Shaft and Tunnels

A five-compartment shaft has been sunk in surface and rock 150 ft. south of the south side of the pit, to a depth of 350 ft. This contains two skips, one cage, one counterweight and one ladder road. The first haulage tunnel put to use this past season was driven at a depth of 290 ft. Another is being driven 60 ft. lower for drainage purposes now and later will be used for second haulage level. A third haulage level will be driven 60 ft. lower for drainage, and will finally be used as the bottom haulage level. Ore then left below this level will be hauled up grade to a point where the tunnel enters the rock side wall of the pit. The incoming water at that time will be taken care of by sinking secondary shafts with drifts tapping the deepest parts of the orebody and installing deep-well centrifugal pumps.

The haulage levels are driven from the shaft to the pit, mostly through rock formation on a +0.5 per cent. grade, running away from the shaft. From the point where these levels break out into the side wall of the pit, it is planned to operate on not to exceed 3 per cent. grade on the main haulage track. With an interval of only 60 ft. between the levels and a pit over ½ mile long, the great bulk of the ore will be moved on much easier grades than 3 per cent.

When the ore level between two haulage levels has been lowered to a point somewhat over halfway between these two levels, a thorough cut will be made from the bottom of the pit to intercept the lower level, and a considerable tonnage of ore between the haulage levels can then be dropped by gravity to the lower level rather than hauled up grade to reach the higher level.

Pumping Equipment

Pumping equipment previous to this past season consisted of two 2000 g.p.m. De Laval centrifugal pumps each direct connected to a 250-hp. 2200-volt slip-ring motor. This plant is at the bottom of a drainage shaft 316 ft. deep. The shaft has a steel headframe sufficiently high and strong to handle all rock materials excavated from pump shaft and pump rooms and handle all heavy pump machinery. In sinking the new hoisting shaft below the level of the existing pump station, two 2500 g.p.m. Layne Bowler deep-well centrifugal pumps were installed in the new shaft and the discharge from these pumps was connected directly into the suctions of the horizontal centrifugal pumps in the old pump station. The pipe line between the two sets of pumps was laid

in a 600-ft. tunnel between the old pump station and the hoisting shaft and a concrete dam was built, containing the water pipes cast in solidly and equipped with a quickly operated bulkhead door. The motors of the deep-well pumps are 20 ft. above the top of the connecting tunnel for protection in case of heavy runs of water, and the water ends are 50 ft. below the bottom of the drainage tunnel from the pit. If more water runs into the shaft from the pit than the pumps can handle, the door in the dam can be closed and the shaft and pit bottom can be allowed to fill up to a depth of many feet while the pumping plant continues to operate. When the third tunnel is completed the water ends of the deep-well pumps will be lowered to a point 50 ft. below the bottom of the tunnel, leaving the motors in their present position. This arrangement with this type of pump eliminates the necessity of pump stations at each level.

After completion of stripping operation in 1931 incident to widening out the upper portions of the pit, all of the old steam locomotive and haulage equipment will be removed from the pit and only the following equipment will be utilized in the pit operation:

3 Bucyrus 120-B full revolving caterpillar traction electric shovels, 32-ft. booms, dippers of 4 cu. yd. water level capacity, similar to the two now in use at the Mesabi Chief.

4 motorized locomotive cars.

25 trailer cars of 4½ cu. yd. water level capacity; all steel construction; six to be operated in a train with each of the four motorized locomotive cars.

In selecting the loading haulage and hoisting equipment, it was decided to use the dipper capacity of the shovels as a unit; *i. e.*, 4 cu. yd. level full. The new haulage cars described below hold 4½ cu. yd. level full, and the two hoisting skips each hold one carload.

The standard railroad-type cars formerly loaded and hauled out of the pit were replaced during the past season by the following specially designed equipment to handle ore from open pit to shaft.

All of the new haulage equipment is designed for standard gage tracks, 4 ft. 8½ in. It was built by the Differential Steel Car Co. and embodies new features which are incorporated in this installation for the first time in mining practice (Fig. 6). The motorized locomotive car with its train of six trailer cars compared to standard locomotive and seven trailer cars of equal capacity gives not only a saving in weight but reduced rolling friction because of one car less to handle. The saving in weight is due to the fact that when the train is empty the locomotive car is about 8 tons lighter than the empty train. When the locomotive car is loaded with ore, the additional weight will enable it to handle the loaded train properly. In other words, the ballast in this type of locomotive consists of useful load and the dead weight of one car is eliminated from each train. The locomotive car (Fig. 7) shows a reduction over the ordinary

type of train of 18 per cent. in weight when light and 10 per cent. when loaded. This means a considerable direct saving in power consumption. The locomotive car has down-folding doors, hinged to the car body on the bottom, and can be dumped either way with a single air cylinder trunnioned over the center of the car body and mounted on a heavy frame. At one end of the car body is the engineer's cab and at the other end a dummy cab. The car as a whole is of compact design, to enable the use of a tunnel of minimum size. In the engineer's cab are mounted

FIG. 6.—NEW LOCOMOTIVE CAR HAULAGE SYSTEM.
Latest type full revolving electric shovel, caterpillar traction, 4-yd. bucket; also air-controlled 4-yd. dirt cars and new locomotive car developed for this kind of operation.

controllers, control contactors, straight and automatic air-brake valves, side-arm collector operating valves, sanding valves and compressor governor. In the dummy cab at the other end of the car are the air compressor and reservoirs. The height of the cab is 7 ft., or approximately the same as the top of the car body. The locomotive cars are fitted with four 33-in. dia. steel wheels mounted outside the frame, Timken bearings on the axles, with a wheel base of 11 ft., enabling the use of extremely sharp track curvature within narrow areas in the pit. All wheels are air-braked and automatic couplers are provided. All parts of the locomotive car are accessible because the brake cylinders and triple valves are mounted on the outside of the frames. Motors and resistor can be readily inspected or repaired when the car body is in a dumping position. Electric equipment was furnished by the General Electric Co., and consists of two type HM-840, 95-hp., 600-volt traction motors, single and magnetic control for each car; also one type CP-130 50-cu. ft., 600-volt air compressor, complete with control and brake valves, gage panels, and so forth.

Each locomotive car is equipped with two side-arm current collectors, one mounted on either side of the engineer's cab, air-operated and controlled by valves mounted conveniently to the engineer. These collectors are designed to serve a double purpose; namely, to collect current in the pit where the trolley wire is installed 8 ft. from the center of the track, and 8 ft. 6 in. above the top of the rails, and also in the tunnel where the trolley wire is installed 2 ft. from the center of the track and 8 ft. 6 in. above the top of the rails. To accomplish this, each arm is attached to a pivot mechanism, which can be rotated 180° by a rack and pinion motion operated by an air cylinder. A manually operated

FIG. 7.—NEW HAULAGE CARS OF 4 YD. CAPACITY AND NEW LOCOMOTIVE CAR.

stop is provided, however, which prevents the pivot from moving through more than 90°. When the arm is in the "off" position, it is parallel with the track, and the rack in mid-position with the pivot against the stop. For operation in the pit after air is admitted to the cylinder, the arm can only swing outboard to a position at right angles. For operation in the tunnel the stop is disengaged, and when air is admitted the arm swings inboard to a position at right angles. Therefore both arms can be swung outboard at the same time, but only one at a time can be swung inboard, depending on which stop the engineer disengages. A suitable cam device depresses the collector arm when rotating and springs hold it against the trolley wire when in running position.

Haulage Cars

The trailer cars (Fig. 7) have bodies of the same design as the locomotive cars and are air-dumped with similar mechanism. The frames have four 20-in. dia. wheels with Timken bearings in the hubs. Air brakes

are fitted to all the wheels and the cars are equipped with automatic couplers. These cars are heavily built and follow closely the general design of the larger cars built by the Differential Steel Car Co. The first of these larger cars put into use in the Lake Superior district were the five 24-yd. cars installed at this same property. The use of standard-gage tracks and air-dump cars underground has been adopted, for its marked advantage in capacity and safety in operation at the speeds necessary to handle so large a quantity of material in so short a time.

The haulage equipment operates on 600-volt direct current taken from a 300-kw. six-phase synchronous converter. The converter and its control apparatus is in the engine house near the collar of the shaft. The haulage feeder panel is provided with automatic reclosing features. The converter is protected with induction overload relays, and induction phase balance relay, bearing temperature relays, a ground protective relay and a high-speed circuit breaker to protect the converter from flashover.

The underground shaft station is arranged so that two cars can be dumped at a time into a suitable pocket for loading the skips and a run-around track is provided so that trains can be rapidly unloaded and dispatched back to the pit.

Headframe and Surface Layout to Handle Four Grades of Ore

Four separate grades of material have to be handled from shaft, so the headframe and pockets are designed to meet this condition. The track layout necessary to handle the heavy daily production from the pit required four lines of parallel tracks, two on either side of the shaft to hold the empty cars above the shaft and the unloaded cars below. A fifth parallel track is used as a run-around by the railroad to run empty cars in above the shaft and on to a tail track and then back-switch in over ladder track to each of the four other tracks. All of the tracks are on a 1.3 per cent. grade past the shaft, so that empty cars may be dropped by gravity to a point under the loading pockets and then dropped again by gravity down into the storage yard after loading. This track arrangement requires that the pockets shall be in a line at right angles with the track system, one pocket being over each track. Both skips discharge into a common hopper, which has a single outlet discharge into a transfer car that runs on a track laid directly over the four pockets and is used to distribute the four kinds of material to the different pockets. This transfer car is operated by a main and tail rope-haulage system, driven by a 7½-hp. motor, which with its control panel is located in a small separate room built especially for this purpose in the headframe, and on a level with the loading platform, which is at sufficient height above the ground, to be used later, when open-pit mining ceases, for a level for stockpiling underground ores hoisted in the winter time. The move-

ment of transfer car from the hopper to any given pocket is controlled by the hoisting engineer through a system of selective push buttons and the car is returned automatically to the loading position under the hopper. No attendant is required in the headframe to distribute the material, as the skiptender in the shaft station underground signals the grade of material direct to the hoisting engineer, who in turn dispatches the material to the car pocket, after it has been dumped into the transfer car. The signal system thus used consists of a small panel mounted at the underground shaft station, on which are installed four jack receptacles. Each jack receptacle controls a small light on the hoist operator's panel. When a certain grade of material is to be hoisted, the skiptender inserts the jack plug in the receptacle corresponding to the grade of material in the skip. This flashes the proper indicating light on the hoist operator's panel. Immediately below each indicating light on this panel there is a jack receptacle, which in turn controls selector relays on the motor control panel in the transfer car. After the skiptender has made his signal and rung up the skip, the hoist operator inserts his jack plug in the receptacle below the light indicating the grade of material, thus setting up the selector relay and establishing proper control and sequence to move the transfer car to the proper pocket for this particular material. After dumping the skip into the transfer car, the hoist operator presses a start button which energizes a master relay and puts the car into motion. The transfer car then moves to the proper pocket, where a limit switch stops it. As soon as the car stops, a timing relay is energized, which after a 10-sec. delay, during which the car has dumped, reverses the motor and returns the car to the shaft hopper, where a second limit switch stops it. When the car is under the hopper at the shaft a green indicating light burns on the hoist operator's panel, showing that the car is in a position to receive ore. When the car is away from the shaft hopper, the green light does not burn. The tripping cams at the two pockets remote from the shaft are stationary and the tripping cams at the two pockets adjacent to the shaft are pivoted so that when depressed they allow the car to move to the outside pockets without moving the load. When the selector relays are set for the pockets adjacent to the shaft, solenoid valves are energized and admit air to cylinders which raise the cams into position to trip. The selector relays thus control the limit switch and trip cam operation. The time of the car cycle from the shaft to the farthest pocket and return, including dumping time, is about 20 sec. faster than the time of the skip from underground station pocket to the surface dump. Fig. 8 shows the equipment described.

The headframe transfer car is novel in design; it somewhat resembles a clam-shell bucket, mounted on a heavy frame with four 20-in. dia. wheels, fitted with Timken bearings in the hubs. Each half swings on trunnions which are so located that when the car is empty the center of

gravity of each half of the car body is outside the trunnion center, thus causing the two halves to remain closed. When loaded, this same condition prevails. When the trip mechanism operates, it tends to open the two valves by rotating them about their trunnion, thus changing the location of the center of gravity, throwing it inside the trunnion points; the load itself then opens the lips, spilling the ore through the car bottom. Two pockets on one side of the shaft are used for Bessemer and non-Bessemer ore, while the two pockets on the other side of the shaft are used for screened ore and the rock screened out of the ore, respectively. A shaking screen is provided over the screened ore pocket, so arranged that the undersize falls directly into the pocket and the oversize discharges into the rock pocket.

FIG. 8.—HEADFRAME AND HOIST HOUSE. TRANSFER CAR IN DUMPING POSITION ABOVE ORE POCKET OVER SHAFT TRACK.

The ore hoist, designed to handle one 8-ton skip per minute, consists of a cylindroconical type of drum, driven through herringbone gears by a 500-hp., 360-r.p.m., 2200-volt slip-ring induction motor. Two skips of 8 tons capacity are hoisted in balance, using 1½-in. rope. The cage hoist for handling men and materials is driven through herringbone gears by 150-hp., 2200-volt slip-ring motor. These hoists were built by the Ottumwa Iron Works and the motor and controls by the General Electric Co. The controls are full magnetic and use time limit acceleration. Skips and head sheaves were built by the Lake Shore Engine Works.

Power for the entire operation of this property is purchased from the Minnesota Power and Light Co. at 22,000 volts, three-phase, 60 cycles, and is stepped down to 2200 volts at the mining company's transformer station.

As mining operations progress to deeper levels in some of the other open-pit ore properties on the Mesabi Range, which are so limited in area that haulage tracks of reasonable length and grades can not be utilized because of increased costs, it is probable that installations similar to that recently put in at the Susquehanna property will be made, providing the open-pit tonnage remaining is great enough to justify the capital expense involved.

DISCUSSION

C. F. Jackson, Washington, D. C.—I had the privilege of working with Mr. Hunner for five years. I am familiar with some of the earlier developments of the company; and I also worked for five years with R. S. Walker, who is consulting engineer to the company, and who is rather a genius as regards electric equipment. He has been a pioneer, and while I was not directly connected with the development of this electrical equipment, I was in quite frequent contact with it through Mr. Walker. I know he had a great deal to do with the development of the electric shovel, in cooperation with the manufacturers of electric equipment, and the engineers of the Bucyrus and Marion companies.

At the Susquehanna pit, one of the especial reasons for not employing the usual method was that a number of rock inclusions occur rather irregularly through the orebody, and all that material would have to be milled down and screened later on. With the present method the rock and ore can be kept separate. Another reason was that large chunks of ore often cause the raises to plug where the milling method is used.

E. Moldenke, Windsor, N. S.—What was the reason for changing from a 4-yd. shovel to a 5-yd. shovel? It is a rather radical change, from 120-B type to 175-B type. The two machines vary considerably in size, weight and mobility.

G. S. Rice, Washington D. C.—In visiting a deep brown-coal pit near Cologne, Germany, I was interested in the latest developments of transportation: the use of a cogwheel electrically driven locomotive to haul a train of large cars out of the pit. In the pit the trains were hauled by electric trolley locomotives. Tracks were standard gage; arrangements for shifting the car tracks were made mechanically by running a special car with vertical rollers; the hangers for the trolley wires were bolted directly on cross stringers, which ran under each track, so that track and trolley were shifted together. This pit is one of the largest in depth, size and capacity in the world. It has a capacity of about 24,000 tons per 24 hours.

H. A. Coy, Mascot, Tenn.—I was interested in the headframe design and the use of transfer car. What conditions were involved that made this method of distributing ore from skip to railroad car desirable?

E. E. Hunner.—In answer to Mr. Moldenke's query the users of the 5-yd. shovel say that although it is considerably heavier than the 4-yd. shovel it is practically as portable—that is, it will go almost anywhere a 4-yd. shovel will go—and is much more heavily powered and therefore much quicker on the swing. It can load faster than the smaller shovel because of the greater quickness on the swing and the greater capacity of bucket. The engineers of the principal user in this field advise me that they are inclined to believe that this is the coming shovel for operating in large orebodies and when the tonnage requirements are high, even taking into account the considerable difference in purchase price.

The reason for the use of the transfer car is that we are hoisting through the Susquehanna shaft four kinds of material that must be kept separate: Bessemer ore, non-Bessemer ore, ore intimately mixed with rock and coarse rock. The rock from the third lot is screened out in the headframe and run over into the fourth pocket, which also receives the coarse rock hoisted from the bottom of the pit, which is encountered as rock horses in the orebody. The screened fines fall into the third pocket and are mixed and shipped with the higher grades of Bessemer ore and non-Bessemer that drop into the first and second pockets, depending on the grade of the screen fines.

I would add to Mr. Jackson's remarks that still another reason for not using the old-fashioned milling system—that is, of milling iron ore down through mills, drawing off this ore from the chutes and tramming through underground haulage to shaft—is that the Susquehanna ore is sticky and the ore would hang up in the chutes. This really would have given even more trouble than trying to mill the rock that he speaks of. Dumping two or three cars through the underground station shaft pocket does not give the difficulty there would be in milling large quantities of ore through a mill hole which might be as much as 60 ft. deep, using the vertical distance that exists between the present haulage levels.

Protective Measures against Gas Hazards at United Verde Mine

By Oscar A. Glaeser,* Jerome, Ariz.

(New York Meeting, February, 1930)

The United Verde Copper Company's mine is at Jerome, Ariz. The orebodies are of the schist replacement type, the main sulfide mass being a large lens-shaped body approximately 7 acres in cross-sectional area. In general, the mineralization is found along the iron schist contact, with the orebodies extending about 1000 ft. along this contact and varying from a few feet to 250 ft. in width. Three types of ore are mined; schist porphyry and the so-called "heavy" or "massive sulfide" ore. Approximately 50 per cent. of the total tonnage mined at present is of the latter class. A representative analysis of this ore indicates 42 to 46 per cent. sulfur.

Because of the high sulfur content, blasting in this ore is an extremely hazardous operation. Not only does it endanger the lives of men who may be in the mine at blasting time but it also becomes a potential fire menace when timber is in close proximity to shots being fired in these massive sulfides. The origin of the fires of 38 years ago, embers of which are glowing to this day, may perhaps be attributed to dust explosions due to blasting in the massive sulfides of those early square-set stopes.

Dust Explosions

Dust explosions in coal mines are a common source of mine disasters. That such explosions may be of common occurrence in metal mines is not generally known, but it is known that metal-mine dust will explode and such explosions have been investigated by the U. S. Bureau of Mines at the Pittsburgh Experiment Station. Samples of "heavy sulfide" ore from the United Verde mine were used in the tests. After studying the results of the tests it was concluded[1] that:

1. Dust explosions were initiated in sulfide ore dust in the gallery with a charge of as low as 75 g. of 60 per cent. gelatin dynamite.
2. The exploding dust generated considerable pressure.

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* Safety and Ventilation Engineer, United Verde Copper Co.

[1] E. D. Gardner and E. Stein: Explosibility of Sulphide Dust in Metal Mines. U. S. Bur. Mines *Rept. of Investigations* No. 2863 (1928).

3. Ignitions were obtained with each of the explosives used.
   a. 60 per cent. ammonia gelatin dynamite,
   b. Permissible gelatin dynamite,
   c. FFF black blasting powder,
   d. 60 per cent. gelatin dynamite,
   e. An ammonia nitrate permissible.

4. There was no "zone of doubt" as to whether ignition did or did not occur. The dust ignited strongly or not at all.

5. The atmosphere contained as much as 2.46 per cent. SO_2 gas after an explosion of sulfide dust in the gallery. The amount of magnetic particles in the dust deposited on the gallery surfaces showed that over 90 per cent. of the particles that had been raised into the exploding cloud had entered into the burning reaction.

6. It took a larger charge of the permissible gelatin than of the 60 per cent. gelatin to ignite the dust.

Conditions Causing Explosions

Exactly what transpires at the face when an explosion occurs is not known, of course. The sulfide ores are extremely hard and tough, and require a relatively heavy charge of explosive. The ore breaks with sharp edges and is heavy, both attributes having a tendency to create dust to a greater degree than with common ores. Furthermore, because of its greater weight the dust settles quickly, thus accumulating in close proximity to the working face. The first few shots to "go off" charge the atmosphere with their own dust and stir up the dust that has accumulated during the shift. Finally some shot with considerable flame ignites this suspended dust and an explosion occurs.

The explosions are always local. They do not propagate and are not of sufficient violence to cause destruction. The gas seems to hang together and to move in a body. It is not easily scattered and dispelled into the air current. It has the appearance of a dense white cloud, and usually fills the entire mine opening.

Dangerous Gas

That the gas is generated in dangerous concentrations is a fact. On several occasions fatalities have occurred when men have tried to fight their way through it. Samples of gases were taken in a heavy sulfide drift 15 min. after blasting and were analyzed by the U. S. Bureau of Mines.[2] The analyses indicated 0.09 per cent. SO_2 and 0.07 per cent. H_2S. A concentration of 0.05 per cent. of SO_2 is dangerous to persons exposed longer than 30 min., while a concentration of 0.06 per cent.

[2] E. D. Gardner, G. W. Jones and J. D. Sullivan: Gases from Blasting in Heavy Sulphides. U. S. Bur. Mines *Rept. of Investigations* No. 2739 (1926).

H_2S will cause unconsciousness within 2 min. and death within 15 minutes.

In order to protect life and property against this ever-present menace, efforts were made to eliminate the cause. Experiments along this line were not entirely successful. Permissible explosives were tried without success, as is also indicated by the Bureau of Mines experiments. Thoroughly wetting down the walls and muck pile before loading and a conscientious effort at tamping have reduced the number of dust explosions to some extent, but not entirely; however, it is believed that the fire hazard is considerably reduced by these precautions. So far it has been customary to use dry tamping previously loaded into paper tamping bags. Wet tamping will be tried soon, and it is believed that the wet clay will prove superior in every respect to the screened dry clay, which is now used.

While these efforts have had a tendency to reduce the hazard, they have not eliminated it. Other means for greater safety had to be found. It was decided to do all blasting under greater regulation and to so control the ventilating air currents that with reasonable care men would find a clear passage to the shaft.

Regulating the Blasting

All heavy blasting is confined to the end of the shift and heavy blasting in the "massive sulfides" is confined to the end of the afternoon shift. Only block hole blasting is permitted during the lunch hour. These regulations assure a clear atmosphere throughout the day, and since the shifts change on the surface, sufficient time elapses between the blasting at the end of the day shift and the beginning of the afternoon shift for the air currents to sweep the mine free of all powder smoke. The graveyard shift is small, consisting chiefly of timber "rustlers," who can easily avoid any local gas condition. Gases are seldom found in the mine when these men go to work; only occasionally is even a trace detected.

Table 1.—*Blasting Schedule*

Spitting Time		Levels
Day Shift	Afternoon Shift	
3:15 p. m. On signals	11:40 p. m.	300 to 2100 incl.
3:20–3:30 p. m.	11:45–11:55 p. m.	2250
3:35–3:45 p. m.	12:00–12:10 a. m.	2400
3:50–4:00 p. m.	12:15–12:25 a. m.	2550
3:50	12:20	2700–3000

A blasting schedule for different sections of the mine and for individual levels has been established. To increase the safety factor, a system of electric blasting signals is in operation from the 2100-ft. level downwards (Fig. 1).

The blasting above the 2100-ft. level is all done at one time. This is possible because of the class of ore mined, the location of the stopes with respect to other workings above and the direction of the flow of air. Below the 2100-ft. level the workings are somewhat more concentrated, more mining is done in the massive sulfides and the flow of any split of air is through several working places.

FIG. 1.—ELECTRIC BLASTING SIGNAL, SHOWING PULL SWITCH, LIGHTS AND HORN.

Beginning with the 2100-ft. level and going downward, the levels are connected by electric signal lines with pull switches and lights on each level. At blasting time the shift boss on the 2100-ft. level sends out word to blast. He stations himself at the blasting signal and checks his men out as they pass him on their way to the station. When all have been checked out past this point he flashes the clearance signal to the 2250-ft. level. The shift boss there receives and returns the signal. This clears the boss on the 2100-ft. level and he goes to the station.

TABLE 2.—*Fresh Air Distribution*

Level	Volume, Cubic Feet	Level	Volume, Cubic Feet
1,000	24,000	2,100	16,500
1,200	11,000	2,250	31,500
1,350	9,500	2,400	40,000
1,500	8,500	2,550	18,000
1,650	16,500	2,700	10,000
1,800	23,000	2,850	7,500
1,950	15,500	3,000	8,500
Total			240,000

The boss on the 2250-ft. level then sends out his order to blast. The checking out process and signalling to the next lower level is then carried out again, as described. Blasting signals at present are used down to the 2550-ft. level and soon will be extended to the 2700-ft. level. Table 1 gives some idea of the time required to carry out a blasting operation.

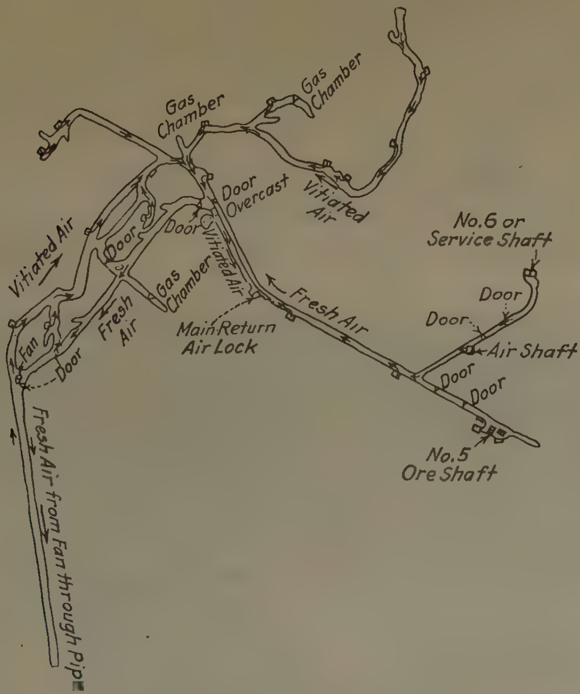

FIG. 2.—PLAN OF 2100-FT. LEVEL.

The actual loss of time at the working face is not great, as approximately the same time interval is required to transport the men to their particular level when going on shift.

VENTILATION

The safety of this system of blasting by electric signals is absolutely dependent on an ascending air current. It can readily be seen what

would happen if one or more raises were downcast; the fumes from the blasting on the level above would come down into the stopes where men were spitting their shots and probably would trap them there. The air current must be so distributed and controlled that its movement will be upward.

Air splits, which were described in an earlier paper,[3] have aided materially in maintaining an ascending air column. It should be understood that all levels are supplied with fresh air (see Table 2) but that certain split levels or gathering levels for return air are so arranged that

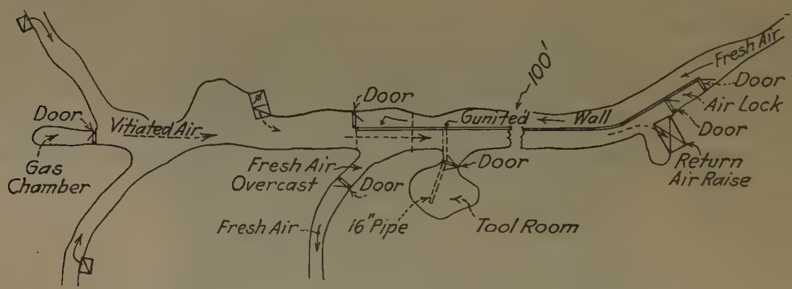

FIG. 3.—DETAILS OF FRESH-AIR BY-PASS AND OVERCAST.

the gathering of the return air and its passage into the main return is separated from the fresh air, which might have to be admitted to the level by means of bulkheads, overcasts or doors, as the case may demand.

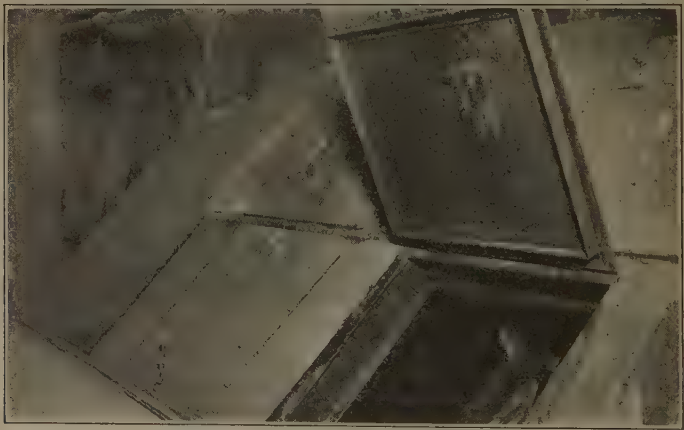

FIG. 4.—CONCRETE RAISE COVER WITH ONE DOOR OPEN.

A sketch plan of the 2100-ft. level is presented in Fig. 2. This is the lowest gathering level in the mine for vitiated or return air. Practically

[3] O. A. Glaeser: Ventilation at the United Verde Mine. *Trans.* A. I. M. E. (1929) 114.

all the air that is admitted on the levels below the 2100-ft. (see Table 2) is removed to the main return of this level. At the same time fresh air in considerable volume is admitted for special use in an outlying district. Fig. 3 is a detailed sketch of the fresh air by-pass and overcast. The tool room, which happens to be on the return-air side, is kept dry by fresh air that is taken into the magazine through a 16-in. metal pipe.

In order to force the return air to leave the workings on this level, it was necessary to effect a seal somewhere between the 2100 and the 1950-ft. levels. A 30-ft. floor pillar was left in place on the 1950-ft.

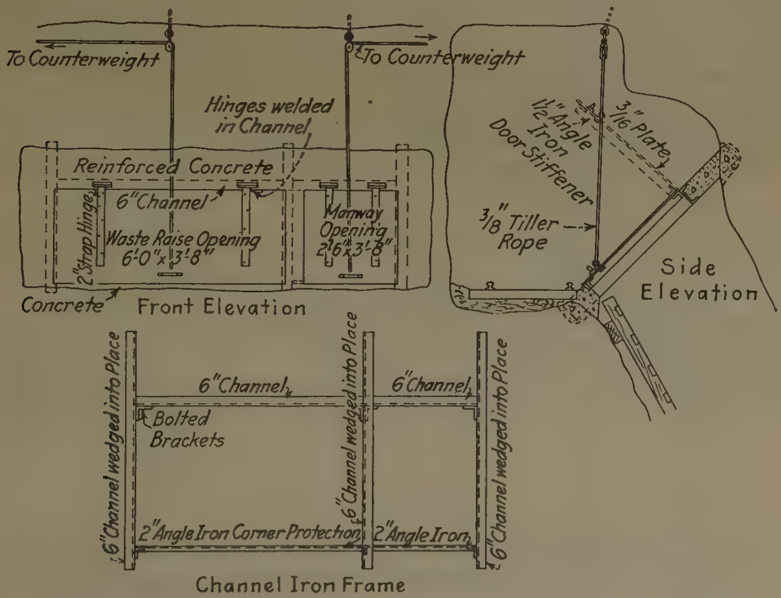

FIG. 5.—RAISE COVER, SHOWING ARRANGEMENT OF DOORS AND SUPPORTS.

level. All raises that were carried through to the 1950-ft. level have been covered by iron doors placed in reenforced concrete collars (Figs. 4 and 5). The floor pillars act as supporting pillars, and with the iron doors in place form a horizontal fire break as well as serving as an air stopping. The cover doors of the raise are kept closed at all times, which assures positive control of the ascending air column on the 2100-ft. level. It further assures a constant supply of fresh air to the levels above, uncontaminated by smoke, gases or vitiated air from the levels below.

SAFETY CHAMBERS

Some years ago gas chambers were necessary for the safety of mine employees, but recently their importance has not been so pronounced.

The present method of blasting and the greater control now exercised over the air currents give reasonable assurance of safe exit. As an additional safety precaution, one or more safety chambers are still maintained on each level, into which men may retreat in any emergency and feel safe as long as the compressed-air lines are intact.

The safety chambers are in dead ends of drifts. A standard-type mine door is installed as far from the face as space outside of the chamber will permit. The chamber is supplied with electric light, water and compressed-air lines. A sign on the door instructs men to close the door and turn on the compressed air. Valves controlling both air and water are inside the chamber. The escaping compressed air builds up sufficient

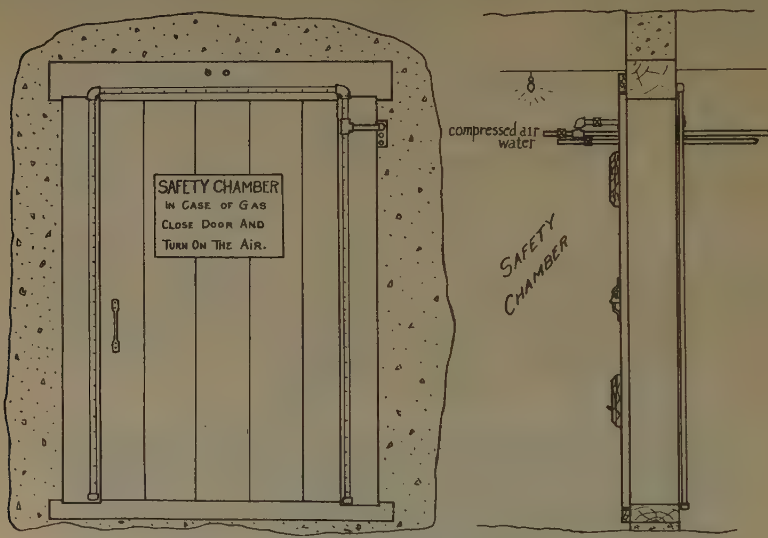

FIG. 6.—SAFETY CHAMBER DOOR, SHOWING PIPE ARRANGEMENT.

pressure within the chamber to keep out all gases under normal conditions. As an additional precaution, a 1-in. perforated air line is fastened on the outside of the door frame along the sides and top (Fig. 6). The pipe is placed so that the perforations will direct the air diagonally across the door opening. It is assumed that this flow of air will not only aid in keeping gases from the door but will tend to dilute any approaching gas. This pipe is connected to the compressed-air line inside the safety chamber. The flow of compressed air through it is controlled by a separate valve.

Signs at various points on the level indicate the direction to the nearest safety chamber, and a green light is placed at the drift intersections leading to the chambers so that men can readily locate them.

Safety chambers should also be useful if men are trapped by a fire. They have never been needed for this purpose in the United Verde mine,

but they should be as effective as a barricade. When used for such an emergency, however, not too much dependence should be placed in the compressed-air supply. If the air line passes through the fire zone, it will soon be burned out or broken, and the air supply would stop, of course. In mines where safety chambers are maintained for this purpose, the piping should be so arranged that the compressed-air will flow to the safety chamber from at least two different sections of the mine, to assure, as far as possible, a continuous supply.

Fire Watchmen

The United Verde mine is patrolled by fire watchmen who have no other duties. They are given a definite beat to cover. Stations have been established where they leave a card on which is written the date, time, condition of that particular district, and their initials. These cards are picked up several times a week and filed away. The watchmen are required to report to the Safety Department daily.

Results of Safety Measures

Blasting signals were installed in August, 1926, and a definite system of ventilation has been established since that time. Rigid adherence to blasting regulations and constant attention to ventilation are the contributing factors largely responsible for the fact that there has been no recurrence of gas accidents since August of 1926.

DISCUSSION

R. S. Lewis, Salt Lake City (written discussion).—The explosiveness of finely pulverized sulfides, as mentioned in Mr. Glaeser's paper, serves to emphasize the fact that many dusts are potential sources of dangerous explosions when they are dry and finely pulverized. The danger from fine bituminous coal dust is well known, but disastrous explosions have been caused by dust from grain, soap, glue, sugar, starch, fertilizer, powdered milk, spice, bark, sulfur, rubber, cocoa and cork. Metallic dusts, such as aluminum and magnesium, have also caused explosions. It would seem that with the exception of inert dusts, like rock dust, almost any industrial dust may cause an explosion provided the dust is fine enough and becomes mixed with the proper amount of air and is then brought in contact with a source of heat sufficient to raise the dust to the ignition point. Anthracite dust prevents the making of a general statement that all carbonaceous dusts are explosive, since it is not normally explosive.

Several years ago, when visiting a coal mine where something like 1,000,000 cu. ft. of methane was given off daily from the coal, one of the striking features of the mining operation noted by the writer was the care taken to prevent an explosion. The safety organization was keenly alert to the danger, and no explosion has occurred at that mine. The safety measures described by Mr. Glaeser show a successful effort made by one metal-mining company to minimize its gas hazards. Many of the dangers of mining can be handled effectively by an efficient safety organization.

B. F. Tillson, Franklin, N. J., told of the fire patrol at the mine with which he is connected. Immediately after blasting, the patrol covers the mine thoroughly, the patrolman carrying a conventional watchman's clock. Stations are established in underground working places close enough to assume complete protection against unseen fires. The men have been educated to the danger of smoking underground and now cooperate readily in enforcing the no-smoking rule. The men are not inspected before they enter the mine, such inspections being considered futile.

D. Harrington, Washington, D. C., felt sceptical about this, pointing out the greater seriousness of an infraction of the rule in a gassy coal mine than in a metal mine, and saying that the recent Oklahoma explosion is said to have been caused by smoking.

S. P. Howell, Washington, D. C., added that in one major coal-mine explosion, where the management was certain no smoking materials went underground, some were found on some of the men.

B. F. Tillson expressed the opinion that legislation or inspection in such matters must always be ineffective and that policing produces the wrong psychology in the men. Success depends on their cooperation and spirit of fair play, his men now being thoroughly sold on the idea.

D. Harrington admitted that it is a hard nut to crack, and cited the case of a Colorado explosion prior to which the men had been searched on going into the mine but after which matches and cigarettes were found in the shoes of one of the men.

C. T. du Rell, Washington, D. C., in support of Mr. Tillson's position of depending on the sportsmanship of the men, said that he felt that one of the gassiest mines in the United States is one of the safest because of a spirit of cooperation which has been engendered among the men.

Development and Installation of the Hawkesworth Detachable Bit*

By Chauncey L. Berrien,† Butte, Mont.

The Hawkesworth detachable drill steel shank and bit were invented by A. L. Hawkesworth, while he was a mechanical foreman for the Anaconda Copper Mining Co., at Butte, Mont. Mr. Hawkesworth died on June 16, 1925, at the time that his invention was practically a proved success. The first application for patents was made by him in November, 1918, and the bit is now being manufactured under patents issued in 1922 and 1923.

The credit for its development in the early stages belongs to A. L. Hawkesworth, Roy S. Alley, Harry A. Gallwey, E. J. Bowman and various officials of the Anaconda Copper Mining Co. Later improvements, manufacturing problems and the direct usage in the mines were originated and perfected by the mechanical department under C. D. Woodward, E. R. Borcherdt, head of the Rock Drill Equipment Department, Robert E. Kelly of the Mechanical Department, and various operating department heads of the Anaconda Copper Mining Co. The author wishes to acknowledge the assistance of these officials in the preparation of this paper.

Comparison of Costs Using Regular Drill Steel and Detachable Drill Steel

In 1922 production was satisfactory and sufficient enough to warrant a complete installation in an operating mine. The study and tests were made at the Badger State mine, the production of which averaged 1200 tons of ore per day. E. R. Borcherdt, head of the rock drill equipment department, was in direct charge of the trials, installation, operating problems and records during the entire period, and still continues in that position.

This mine is a typical Butte mine, operated through a standard four-compartment shaft, served by a double drum hoist and an auxiliary hoist. The texture of the ground and ore has all the variations from soft to very hard granite and sulfide ore with production from levels between the

* Presented at Meeting of American Mining Congress, Spokane, Wash., October, 1929.

† General Superintendent of Mines, Anaconda Copper Mining Co.

1500 and 3400 elevations. Work was carried on in 85 working places, 46 per cent. of which were square set and rill stopes, 28 per cent. levels and 26 per cent. raises. The mine employed 700 men per day at that time, and aside from hoisting ore it also transferred, from level to level, waste filling from development. Like all Butte mines, it was kept up to the capacity of the operating equipment of the mine.

Any mine operator knows that a direct comparison of regular drill steel with detachable steel would not suffice to determine the saving possible, but that many other factors must be considered. It was first necessary to obtain accurate costs of all kinds for regular drill steel. Few data on the cost of *regular steel* were available, so that extensive time studies of the various steps in its use had to be undertaken. These considered the costs of the following factors:

Equipment Cost

Cost per piece of steel delivered at the mine.
Average weight per steel.
Cost per pound.
Cost of fabrication per piece of steel.
Average number of pieces of steel in service at mine.
Total cost of average steel equipment.

Operating Costs

Distribution.—Cost of complete cycles from shop to mine and return to shop. This considered: (1) hoisting and shaft maintenance; (2) hoisting and lowering cost; (3) time of miners in transporting steel from station to working place; (4) nippers' time in handling sharp and dull steel in mine.

Sharpening Drill Steel.—Labor and supply cost. Labor costs included: Sharpening, tempering, reshanking, cleaning plugged steel, straightening bent steel, loading truck for transportation to shaft, delays. Maintenance costs included: Compressed air, coke, lubricating oil, quenching oil, water, sharpener repair, die and dolly upkeep, furnace upkeep.

Replacement Steel Costs.—The costs of fabrication of new regular drill steel and of sharpening dull steel were based on accurate time studies over extended periods. Amounts of steel in service at a mine were determined by inventories of the number of pieces and weights. Losses and replacements were also determined from these inventories. But by far the most difficult determination was that of distribution cost. Time studies of the distribution of steel to each individual working place in the mine required weeks of observation by engineers. A volume could be written covering the details of these observations and results.

With the record of the cost of operation of regular drill steel over a period of one year completed, it now became necessary to remove all regular drill steel from the mine and install detachable equipment throughout. This equipment consisted of shanks, bits, knock-off blocks and carriers, which will be described later. After a lapse of time to permit the miners to become familiar with the use of the detachable bit and to permit the working out of distribution practices, a year's observation of costs was begun.

This involved the determination of the following factors:

Installation Cost

Cost per shank delivered at mine.
Average number of shanks required to equip mine.
Total cost of average shank equipment.
Cost of average bit.
Average number of bits required on hand.
Total cost of average bit equipment.
Cost of knock-off blocks.
Average number of knock-off blocks required.
Total cost of average knock-off block equipment.
Cost of average carrier.
Average number of carriers required.
Total cost of carrier equipment.
Grand total of detachable equipment required.

Operating Costs

Bit cost.
Shank repair cost.
Shank replacement cost.
Knock-off block replacement cost.
Carrier replacement cost.
Distribution cost.

These data were obtained from time studies and accurate records of material used. The detailed records are not presented in this paper because they are so voluminous and could not be used directly by other operators for comparative purposes. At the conclusion of the paper there is given a comparison of costs between regular and detachable drill steel resulting from an all mines installation.

With the complete costs of one year's operation of one mine for both regular and detachable steel at hand it was now possible to make a definite comparison of the credits and debits to each type and also to determine what manufacturing costs must be met to justify the installation of the detachable bit.

Comparative Statistics: Regular and Detachable Steel

The following data on detachable and regular steel were obtained from operating records of two consecutive years at the Badger State mine.

Distribution Comparison

The continuous operation of 85 working places for one year, divided into 39 stopes, 24 sills and 22 raises, required an average installation of 4020 pieces of regular drill steel averaging 15.29 lb. each, or a total of 61,479 lb. Quarter octagon hollow drill steel of 1 in. dia. was used for both stopers and drifters. During this period there were 148,642 pieces of drill steel sharpened, each piece averaging 15.293 lb., making the total weight of sharp steel handled 2,275,182 lb. Doubling this amount for weight of drill steel handled gives a grand total of 4,546,364 pounds.

For the same amount of work done, adjusted to cubic footage excavated and number of working places, the following weight of detachable equipment was required to be handled: 9228 damaged shanks, 8975 repaired shanks and 1223 new replacement shanks, or a total weight of 291,112 pounds. To this is added a total weight of 586,074 lb. of bits and carriers taken in and out of the mine by miners or a grand total of 877,186 lb. of detachable equipment requiring transportation in and out of the mine. This amount is but 19.294 per cent. of the weight of regular drill steel required to be handled. When distribution of shanks only is considered, the weight requiring handling is but 6.403 per cent. of the weight of regular steel. This difference in quantities requiring distribution is, of course, one of the principal attributes which can be capitalized. The value of this reduction depends entirely on conditions and will vary in each installation. The maximum advantage is obtained in difficultly accessible working places; the mimimum in easily accessible places, such as drifts, where a large number of regular steel can be brought in on a truck.

Installation Comparison

Installation of 802 shanks divided into 228 starters, 211 seconds, 199 thirds, 131 fourths and 33 fifths, averaging 18 lb. each, or a total of 14,436 lb., was required to replace 61,479 pounds of regular drill steel. This is but 23 per cent. of the weight of the 4020 pieces of regular steel required for the same work.

Comparison of Drilling Efficiency

Comparison of excavation per regular steel sharpened and per detachable bit dulled, in each case over a period of one year at the Badger State mine, was found to be as follows: regular steel, 30.03 cu. ft.; detachable bit, 25.63 cu. feet.

Increase in detachable requirements over regular steel is 17.2 per cent. Excavation per detachable bit is 85 per cent. of the excavation of regular steel.

This decreased excavation per detachable bit is not indicative of actual decreased drilling capacity, but is rather due to failure of the miners to utilize full drilling capacity of the bit. In other words, on account of the greater labor involved in getting a regular steel to the drilling face, the miner will actually use a regular steel to greater dullness even though it be inefficient to do so. The labor entailed in changing a bit is so small that rather than use a bit to a point of gage loss, where possibilities of sticking the steel exist, he will not take a chance on using a bit a second, third or fourth time. As a matter of fact, we do not recommend using a bit to a point of extreme dullness, on account of the greater vibrational and torsional strains set up in the shank. Vibration is largely absorbed as long as the cutting edges penetrate the rock which absorbs the blow. There is a large variation in cubic feet excavated per regular steel at various mines of the Anaconda Copper Mining Co., ranging from 20 to 80, depending on local ground conditions. While an actual comparison of drilling speed with the detachable and the regular results in slightly higher speed with the regular, no particular difference can be noticed in over-all drilling speed.

Comparison of Replacement Steel

Considerable variation exists in the amount of regular and detachable steel required for replacement. This is dependent on types of mining prosecuted and on types of steel used. For example: At the Badger State mine, where 1-in. quarter octagon steel was used for both drifting and stoping, 94.019 lb. of regular drill steel per working day was required for replacement. At the Bell-Diamond mine, by later trials, where 1-in. quarter octagon steel was used for stoping and 1¼-in. round steel for drifting, the daily replacement required was 138.78 lb. On a basis of excavation per pound of steel consumed, the Badger State excavated 153 cu. ft. per pound of steel as against 91 cu. ft. per pound of steel consumed at the Bell-Diamond. Replacement of regular steel is largely necessitated by loss in the working places, while replacement of detachable shanks is necessitated by breakage or damage. The loss of regular steel is high on account of the large number of steels involved and breakage is low on account of steel being lost before the fatigue limit of the steel is reached. On the other hand, actual loss of detachable shanks is reduced because of the small number of shanks required in each working place, and breakage is increased by the increased duty of each shank. To further illustrate this point, we determined that the cycle of use of each regular steel was such that it was drilled with but once in each period

of a little over two weeks, whereas the detachable shank is drilled with continuously until it breaks or becomes damaged. The actual drilling life of a shank varies considerably, depending upon the nature of the ground drilled. In experimental drilling we have obtained over 300 ft. per shank. It is impracticable to check footage drilled per shank in regular operations, so the average cubic footage excavated is used as an index. The relation of the weight of replacement shanks to the weight of installation shanks for the operation in one mine for one year was approximately 1.5, or, in a mine requiring 1000 installation shanks, 1500 replacement shanks would be required in a year's operation of that mine. The consumption of drill steel for replacement of detachable shanks varies from 50 to 80 per cent. of the amount required for the replacement of regular steel.

Material Required for Detachable Installation

The Anaconda Copper Mining Co., has standardized on 1-in. quarter octagon drill steel for all stopes, drifts and raises, except for sinking and plugging. The advantages of the use of standard section for stoping, drifting, and raising are: (1) convenience of interchangeability; (2) simplifies distribution; (3) requires less stock; (4) reduces loss; (5) lighter weight—easier to handle and less weight per shank.

Table 1.—*Shanks Used*

Section used—1-in. qtr., oct., 0.90 c....................	Stopes Raises Drifts—XC
⅞-in. hex....................................	Sinking Plugging

Length of Shanks

	1-In. Quarter Octagon[a]			1-In. Hexagon	
	Maximum, Inches	Minimum, Inches	Approx. Weight, Lb.	For Plugging, Maximum, Inches[b]	For Sinking
Starters.......	38–34	34–30	10.5	36	Same as 1-in. quarter octagon, changes, 18 in.
Seconds.......	56–52	52–48	15	54	
Thirds........	74–70	70–66	19.5		
Fourths.......	92–88	88–84	24		
Fifths........	110–106	106–102	28.5		

[a] 2-in. tolerance in length over and under standards.
[b] Starter upset for No. 4 bit; second upset for No. 5 bit.

Shank Upsets

The contact surface between bit and shank must naturally vary as the bits decrease in size in order to allow for clearance. At present the

diameters of the upset on shanks for use with starter bits are 1$\frac{1}{2}$ in.; for second and third bits, 1$\frac{7}{16}$ in.; for fourth bits, 1$\frac{5}{16}$ in.; and for fifth bits, 1$\frac{1}{4}$ inch.

Bits

Four-point double-taper cross bits are used, 5° to 14°. Horizontal taper of tongue is 0.75 in. per foot of included angle, and, vertical taper of tongue and grooves, 3°. Angular taper of tongue is 14°. Starter, 1$\frac{7}{8}$ in.; second, 1$\frac{3}{4}$ in.; third, 1$\frac{5}{8}$ in.; fourth, 1$\frac{1}{2}$ in.; and fifth, 1$\frac{3}{8}$ in. Bit size limits are not held closely on account of excess grinding required. Bit gages and shank lengths vary slightly at the different mines. If a bit can be cleaned up by removing $\frac{1}{16}$ in. dia. it is passed.

Bit Carriers

Welded carriers of No. 12 gage sheet steel are made up in either four or five compartments, depending on number of bits used. Several

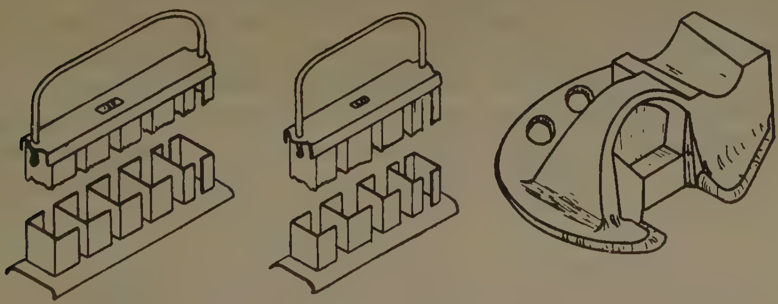

FIG. 1.—STANDARD CARRIERS AND KNOCK-OFF BLOCK.
a and *b*, carriers, *c*, knock-off block.

different types have been used and discarded in favor of the present carrier, holding 20 bits. In early tests bits were drilled to permit them to be strung on wires.

Knock-off Blocks

Blocks for knocking bits on and off have been improved from time to time. The first block consisted of a flat, round iron casting, raised in the center to support the shank when driving the bit on or off. Later improvements were the addition of a hood to reduce losses by retaining the bit when it was detached from the shank, and the addition of a shoulder to support the bit to permit its removal by a blow on the shank. Driving a bit on a shank with an ax is ruinous to the poll, so that it was necessary to develop a tool which has a less damaging effect on the ax and which will insure a better seat between the bit and the shank.

Surface Bit House

A building approximately 10 by 16 ft., situated preferably in line between the shaft and the change house, is required for bit and shank storage and carrier issuance. Carrier racks, shank racks and bins for various sizes of sharp and dull bits are housed in this building.

Underground Shank Lockers

Lockers to house reserve shanks are required on each operating level.

INSTALLATION REQUIREMENTS

The determination of amounts of equipment required for most efficient operation of various classes of working places was made from periodical inventories of material in use in the mine under observation. For example, a mine having 80 working places, divided into 39 stopes, 20 sills, and 21 raises, requires not only shanks in actual use in these places but also a stock of replacement shanks on each working level. In addition, a reserve stock must be maintained in the surface bit house to take care of increases in working places and to provide for the time cycle required for repair of damaged shanks. It has been determined that the number of shanks required for each stope and each sill are the same and that each raise requires three times this number. Considering a stope or sill working place as one unit and a raise three units, we have:

39 stopes	39 units
20 sills	20 units
21 raises	63 units
80 places	122 total units

The average number of shanks for efficient operation of these 80 working places was found to be as follows:

Starters	228, or 1.88 per unit
Seconds	211, or 1.74 per unit
Thirds	199, or 1.64 per unit
Fourths	131, or 1.08 per unit
Fifths	33, or 0.33 per unit
Total	802

An average of 1.5 bit carriers and 1.2 knock-off blocks were required per working place.

CONCLUSIONS OF BADGER STATE MINE TEST

The general results and cost comparisons derived from this fully-equipped mine test, after one year of operating with the detachable bit,

were conclusive enough to warrant the same installation at other mines as fast as detachable steel production could be increased.

Therefore the Anaconda company proceeded with the program as noted in the first part of this article until the present plant production capacity had been reached. Throughout the period from 1922 to 1929, the details of manufacture and usage were improved step by step.

Reliable data were obtained throughout this period, but space is not sufficient to record all such information and experience. At the conclusion of this paper, however, will be found a record of operating results obtained with detachable drill steel in 1928 and up to Aug. 1, 1929, at all mines, over which period the methods of manufacturing, repairing and general operating service were fairly well standardized.

General Methods of Service

Installation Procedure

After determining the number of working places, the proper number of shanks and knock-off blocks are bundled together and taken into the working place by the nippers. One level, or one shift boss's beat, is completely equipped before the equipment of another beat is begun. The following day, carriers containing proper number of bits are issued to miners working in these places. This procedure is followed until the mine is completely equipped and all regular drill steel is removed.

On the day preceding installation, the foreman, assistant foreman, shift bosses and nippers are assembled and given instructions in the use of the bit. The importance of precautions to be observed and the routine to be followed is stressed, for upon these men the successful operation of the bit depends.

Operating Practice at Mines

Strict control of detachable equipment and control of the actual use of this equipment is essential, so that routine of handling and use in each mine had to be standardized.

All carriers are numbered and contain various sizes and quantities of bits required for one shift of drilling under various conditions. They are issued to the miners by the bit-house attendant at the beginning of the shift. The miner gives his contract number so that a record of the issue of the carrier and the number of bits issued can be made. The number of carriers and the number of bits required is determined by the miner. Raisemen will require more than one carrier containing the maximum number of bits, while a stope miner will perhaps require only a single carrier containing a minimum number of bits. All carriers are required to be returned to the bit house at the end of the shift, at which

time a record of the number of bits dulled, damaged, and lost is made as the carrier is refilled. This is necessary to eliminate the loss of the carriers in the mine and to supply a record of the daily loss of bits in individual contracts.

When a miner breaks or damages a shank he is required to exchange this shank for a new one, obtained from the underground locker on that level. This practice is necessary to check the loss or extravagant use of shanks. It is the duty of the nipper to maintain the stock of new or repaired shanks in underground lockers and to return damaged ones to the surface bit house. The bit-house attendant inspects these shanks and records repairs necessary on their return to the shop. Each day the shop is advised of the number and size of the shanks to be returned for repair so that an equivalent number of repaired shanks are delivered to the mine when the damaged shanks are collected. This plan insures control of supply of shanks to eliminate surplus and shortage. When the number of working places is increased, additional shanks are supplied. Daily deliveries of sharp bits and return of dull bits for regrinding are made.

Bit requirements vary with each installation, as does the consumption of sizes. Theoretically, only new starter bits would be required. These starters would be dulled and reground to seconds, which in turn would be dulled and reground to thirds, fourths, and fifths. However, actual records show this not to be the case. In the use of 1000 bits, the percentage of sizes used is shown in Table 2.

A surplus of third, fourth, and fifth bits results when operation is confined to one mine. However, with several mines operating under various drilling conditions, it is possible to reduce this surplus by the allocation of various bit sizes. For example: Several mines use second bits on starter shanks, third bits on second shanks, etc., and all mines use fourth and fifth bits on hexagon steel for plugging. The upset on the shank limits the size of the bit possible to use, and it may prove practicable to use smaller upsets to permit the use of third bits on special starter shanks in the mines in softer ground. Experiments are being carried on with shanks to endeavor to decrease the size of upsets from 4 to 2. If this proves successful the use of 1⅞-in. starter bits may be eliminated and all bits may be made up from the smaller section of cruciform steel.

Loss of bits is largely dependent on the supervision by the shift boss. Some losses can not be eliminated as bits will occasionally be lost in the hole or will fall into chutes where they can not be recovered. Some of these bits are picked up later by the magnets at the concentrator. However, some miners are careless and fail to drive the bit on the shank properly, so that it is likely to come off in collaring a hole. It is believed that the greatest percentage of loss is caused by the miner failing to

return dull bits to the carrier. The percentage of bits lost to the number used varies from 8 to 17 per cent., under the various operating conditions.

TABLE 2.—*Records of Use of* 1000 *Bits*

	Firsts, Number	Seconds, Number	Thirds, Number	Fourths, Number	Fifths, Number	Total Number
Consumption	288.932	293.659	241.663	140.144	35.602	1000.000
Loss	23.370	22.840	24.136	18.425	3.515	92.286
Returned to surface	265.562	270.819	217.527	121.719	32.087	907.714
Damaged	4.765	4.884	4.214	2.513	.957	17.333
Returned for regrinding	260.797	265.935	213.313	119.206	31.130	890.381

RETURNED FOR REGRINDING

Firsts to Seconds	Seconds to Thirds	Thirds to Fourths	Fourths to Fifths
96.445	98.902	97.802	100.
Firsts to Thirds	**Seconds to Fourths**	**Thirds to Fifths**	
3.554	1.098	2.198	

RESULTS OF REGRINDING

	Firsts	Seconds	Thirds	Fourths	Fifths	Total
	....	251.527	9.270	2.920	4.689	
	....	....	263.015	208.624	119.206	
	....	251.527	272.285	211.544	123.895	
Consumption	288.932	293.659	241.663	140.144	35.602	1000.000
Excess reground bits	....	....	30.622	71.400	88.293	
New bits to purchase	288.932	42.132				

New, 30 to 40 per cent.; reground, 60 to 70 per cent.

PLANT STOCK

When a mine is equipped with Hawkesworth detachable steel all shanks are standard length, starters 3 ft. 2 in., seconds 4 ft. 8 in., thirds 6 ft. 2 in., fourths 7 ft. 8 in., and fifths 9 ft. 2 in. As the shanks are damaged in operation they are returned to the Hawkesworth drill shop for repairs.

In the repair process many shanks are cut to a shorter length than the standard. This has made the sets irregular in length and has caused a great deal of trouble to the miners in drilling their round of holes, therefore some mines have changed from a standard length shank to a special length shank, starters 2 ft. 10 in., seconds 4 ft. 4 in., thirds 5 ft. 10 in., fourths 7 ft. 4 in., and fifths 8 ft. 10 in. At present six mines are using standard length shanks and five mines are using special length shanks.

Before the damaged shanks are returned to the Hawkesworth drill shop they are grouped into standard lengths and special lengths. The shanks of special length are painted with a color assigned that mine. The standard lengths are returned to the mine and the special length shanks are placed in the plant stock warehouse and are sold to other mines, credit being given to the mines selling the shanks.

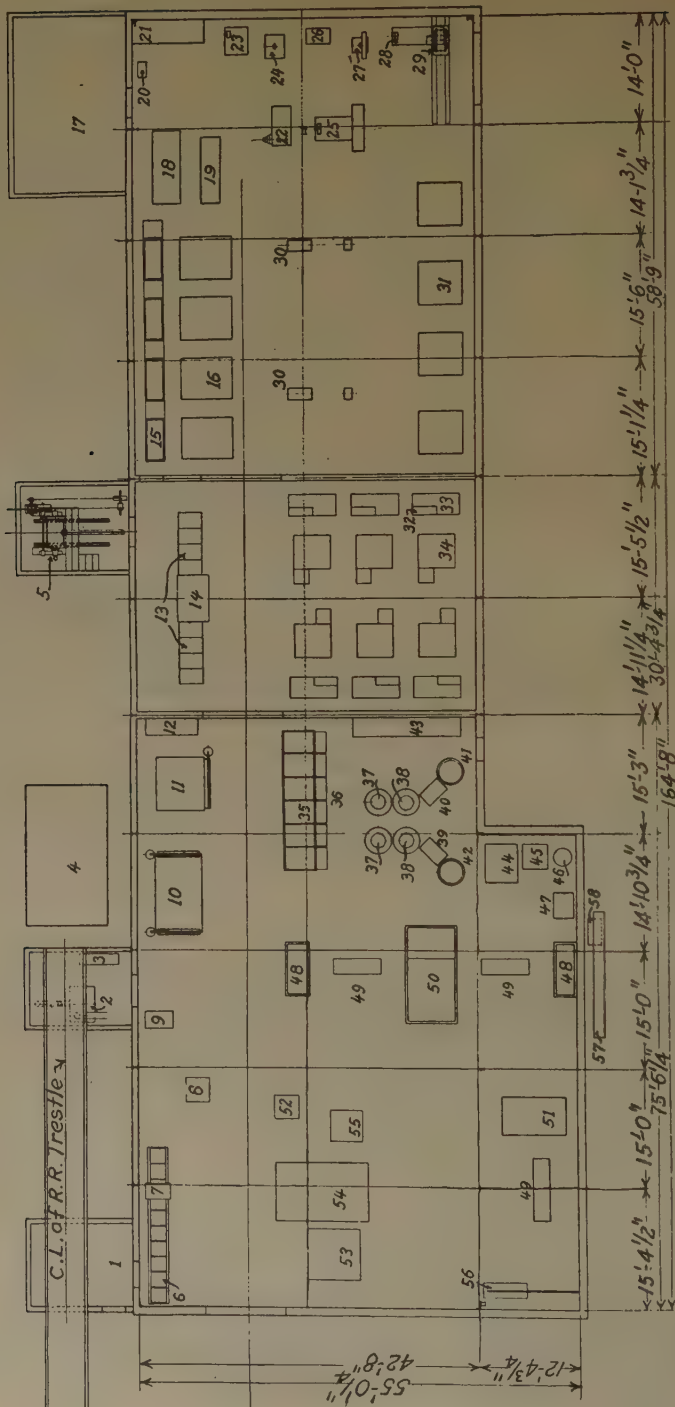

FIG. 2.—General layout of furnace and machine shop building for manufacture of Hawkesworth bits and shanks. Description on next page.

Records of Equipment

Records of Hawkesworth detachable steel are taken care of at each mine by a bit-house attendant. A building or space about 10 by 14 ft. is used as a storeroom for shanks, bits, and carriers. It is also equipped with a bench where carriers are refilled with bits. Deliveries to and from the Hawkesworth plant are made there, miners receive and return carriers of bits, and nippers receive repaired and replacement shanks and return damaged shanks there.

Preferably the bit house is located between the shaft and the dry and where the plant truck can easily make deliveries. A window or half door with a wide ledge serves for miners receiving and returning carriers. New and repaired shanks are stored vertically in a partitioned rack by sizes; starters, seconds, etc. Close to the issuing window are shelves upon which the carriers are kept in numerical order. In about the center of the room is placed the carrier reloading bench. It has two sets of storage bins for sharp and dull bits by sizes. The top set of bins are for sharp bits and the lower bins are provided with a chute at the back for emptying the dull bits into cans to be returned to the plant. A sheet-

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Description of Fig. 2.

1. Brick, clay and oil supply room.
2. Turbo compressor, Spencer Turbine Co.
3. ¾-in. Viking fuel oil pump, with 15-in. dia. × 5-ft. tank.
4. Fuel oil tank—10½-ft. dia. × 18 ft. long.
5. Whiting Corp. 24 × 36-in. style "A" tumbler.
6. Shank measuring table.
7. Oxweld cutting machine.
8. Shank straightener.
9. Shank grinder.
10. Bits—Westinghouse type H-75 electric annealing furnace, 75 kw.
11. Bits—Hoskins "FR-263" electric annealing furnace, 43 kw.
12. Control equipment.
13. Bit bins.
14. Inspection table.
15. Bit bins—capacity of each, 5000 bits.
16. Bit miller.
17. Office.
18. 16-in. Lodge & Shipley lathe.
19. 16-in. Sidney M. T. Co. lathe.
20. Grinder.
21. Bench and vise.
22. Shaper, Whip Machine Tool Co.
23. Vertical miller, Taylor & Fenn Co.
24. Universal grinder, Cincinnati Milling Machine Co.
25. Universal miller, Brown & Sharpe Co.
26. Mandrel stand.
27. Cutter grinder, Pratt & Whitney Co.
28. Drill press, Rockford Drilling Machine Co.
29. 50-hp. Westinghouse motor.
30. Shank-drilling machines.
31. Shank miller.
32. Holder block.
33. Bit bin.
34. Bit grinder.
35. Bit bins.
36. Traying table.
37. 38. Bit hardening, Westinghouse type J, 22-kw. electric furnaces (37, lead pots; 38, salt pots).
39. 40. Cold water tanks.
41. 42. Boiling water tanks.
43. Control equipment.
44. 45. Die furnaces.
46. Die oil tank.
47. Drawing furnace.
48. Lime bins.
49. Shank furnaces.
50. Quenching oil tank.
51. Ajax shank forging machine.
52. Bit trimmer press, Toledo Machine Tool Co.
53. Bit furnace.
54. Ajax bit forging machine.
55. Bit reheating furnace.
56. Shank cleaning machine.
57. Quenching oil cooler, Griscom Russell Co., type M.
58. 1½-in. Bowser quenching oil pump.

steel plate upon the table part of the bench protects the wood. A desk and drawer for keeping records and forms is at one end of the bench. Other equipment includes a gage for gaging dull bits, paint and paint brush for identification of shanks at the plant.

Bits are taken into and out of the mine by the miners using them, in the carriers provided for this purpose. These carriers consist of channel-shaped sheet metal welded to a base, with a channel-shaped cover attached by a hole in each end of the handle, upon which it can slide (Fig. 1). The tubular handle is hinged at each end of the carrier and is so shaped as to allow the cover, which just overlaps the compartments, to be raised so that the handle and cover turn out of the way to give access to the compartments. Four-compartment or five-compartment carriers are used, depending on the number of shank lengths used. If four changes of steel are required in drilling, four sizes of bits are required. Four bits of each size are put into the carriers. A four-compartment (16-bit) carrier weighs 15 lb., and a five (20 bits) 17½ lb. Each carrier is numbered by braizing one end and stamping.

The bit carriers are issued to miners by the bit-house man, who records each carrier number and the contract number of the place to which it is going. All carriers are returned at the end of each shift. As each carrier is refilled for the next shift, the number of bits lost, damaged, and dulled are recorded against its previous issue. Sharp bits are taken from the plant to the bit house, and dull bits to the plant by the plant truck. These are recorded on the delivery forms.

Shanks are taken into and out of the mine by the nippers. Each day all damaged shanks are hoisted, inspected, and bits removed by bit-house attendant, and sent to the plant. The following day these or a like number of shanks are returned to the mine. Replacement and surplus shanks are stored in the bit house and on the larger underground stations.

The bit-house attendant starts to work at 10 a. m. He fills carriers, orders materials, keeps bit and shank records, signs for deliveries, and takes monthly inventories of supplies. He issues the carriers for the night shift, while a boss toolman issues those on the day shift.

In order that accurate detailed records might be kept of all detachable drill steel service and costs, the following system was adopted. The present system became a standard after five years of tests and after the mines were fully equipped.

The original installation at a mine is naturally a definite order for shanks, bits, knock-off blocks, carriers, etc. After that we are concerned only with replacement of equipment and we have provided for a monthly record for the information of all concerned by using the following forms:

Form, Rock Drill 25

This is an order form, the headings of which are:

Bits
- Size, new
- Size, reground

Shanks
- 1-in. quarter octagon
  - Quantity
  - Length
- 7/8-in. hexagon
  - Quantity
  - Length

Carriers

Knock-off blocks

The bit-house man at each mine makes out the order in duplicate, the original is approved by the mine foreman and sent through the rock-drill department to the Hawkesworth drill shop. The duplicate remains at the mine bit house.

Form, Rock Drill 27

This form is used when shanks and bits are delivered to the mine from the drill shop. Its headings are new bits, reground bits, repaired shanks from shop stock and repaired shanks from mine stock. It must be signed by the shop stockkeeper and the bit-house man.

Form, Rock Drill 31

This form is used to keep the record of dull bits and damaged shanks returned from mine to shop. Its headings are size and number of dull bits, lengths and kinds of damaged shanks, damaged carriers and damaged bits. It is signed by the bit-house man.

Form, Rock Drill 30

This form is used to keep an accurate detailed record of all shanks and bits purchased and the dull and damaged bits returned to the shop. This record form is used at the shop and is made out for each mine, indexed and kept in ledger form to facilitate the work of the storekeeper. At the end of each month these forms are forwarded to the rock-drill department, in order to complete the form for Rock Drill 32.

Form, Rock Drill 32

This form being a recapitulation of the forms as noted above, plus repair items at shop, shows, *by mines*, the following: Bits received; reground bits received; shanks received, both new, replacement and repaired, with credit, if any; damaged shank to shop; dull bits to shop;

distribution of bits to stopes, drifts, raises, and all workings with the record in each case for issued, returned, lost, damaged, and dulled. It also has a heading for bits reported used and the cubic feet excavated per bit used in stopes, drifts, raises, and all workings. The excavation information is taken from engineering department records.

This form also has headings showing details of repairs to shanks, such as: Slightly damaged bit end; damaged bit end; bent or blasted, plugged, worn, undersized chuck end; wing of shank broken; reshank chuck end; regrind chuck end; redress broken steel; reshank broken steel; and total shanks repaired.

Other headings are percentage of grand total repairs, cubic feet excavated per shank repaired, total repairs per shank, and dulled bits on damaged shanks.

It also shows carriers received, knock-off blocks received, the percentage loss of bits issued, percentage loss of bits used, an inventory record, and the total shop production for new shanks and in stock.

While these details seem excessive, the records are easily kept and are a necessity in checking up results and enforcing proper usage. The rock-drill department sends this complete report to all superintendents and mine foremen at the end of each month.

Form, Rock Drill 26

This form is used to record the use of bits at the mine, the distribution of bits to the various workings, with a record of the lost, damaged, and dull bits. They are forwarded from each mine to the rock-drill department and the results are used in compiling Form 32. The headings are contract number, carrier number, issued, returned, loss, damaged, and dulled, all classified into stopes, drifts, and raises.

Form, Rock Drill 29

This form is made out daily at the shop and forwarded to the rock-drill department at the end of each week. It is used in making up Form No. 32.

The system of records is standardized to such an extent at this time that it is giving satisfaction and will not require many, if any, changes or additions. Anyone familiar with operating problems in a mine will not require a detailed explanation as to the reasons or necessity for keeping such records.

Mechanical Development and Manufacture

While the first application for patents was made in 1918, Arthur L. Hawkesworth conceived the idea of a detachable drill bit in 1916. During the ensuing eight years tests were conducted at intervals in the

Anaconda company's mines, and the evolution of the bit to its present conformation was the result of the efforts of the inventor, his associates, R. S. Alley and H. A. Gallwey, and the aid and cooperation afforded them by the Anaconda company. Much credit is due to C. D. Woodward and Robert E. Kelly, of the Anaconda company, for the development of manufacturing, and for the presentation of the following details.

Early in the year 1924 the Hawkesworth detachable drill bit was perfected to a point where the installation of a major manufacturing plant seemed advisable. With this end in view the Hawkesworth Drill Co. incorporated and secured three buildings at the Anaconda Copper Mining Company's West Gray Rock mine. Ninety-three hundred square feet of floor space was available, and while a satisfactory flow sheet could not be worked out, on account of physical difficulties, the Hawkesworth company installed machinery and equipment to the best advantage and proceeded to manufacture bits and shanks on a larger scale than had been attempted before. Naturally, vital changes in manufacturing processes and heat treatment occurred during this period. Machines were tried and discarded and special equipment was developed to meet some particular step in the manufacturing process. A satisfactory heat treatment was perfected, which was as important to the success of the bit as any other manufacturing feature.

One phase in the history of the bit during this period is worth recording. Early attempts in forging were not satisfactory, due primarily to the fact that proper forging equipment was not available in Butte. A cast bit seemed to be the solution of the problem, so in order to determine the feasibility of this step 10,000 bits cast from alloy steel were ordered from an eastern manufacturer by the Anaconda company. The result was disappointing. Fully 50 per cent. of the product showed serious casting defects, rendering the bits either unsuitable for drilling or unfit for further grinding. As a consequence of this experience, no further consideration was given to a cast bit.

The manufacturing equipment selected by the Hawkesworth company for the new plant included a screw-forging press for bits, trimming press, oil-annealing furnace, bit-milling machines, pot-hardening furnaces, drill sharpener for upsetting shanks, shank miller, shank-hardening furnace, oil-quenching tanks and various machine-shop tools for die sinking and repairing manufactured equipment.

The capacity of this plant was about 1200 new bits, 60 new shanks, 2500 reground bits, and 120 repaired shanks per 8-hr. day. An average of 14 men were employed. The Hawkesworth company began production in November, 1924, and continually operated until June 16, 1928, when the plant and limited patent rights were acquired by the Anaconda Copper Mining Co. Manufacturing methods introduced at that time are essentially those employed today.

During the early period of the bit development, alloy steel was used almost exclusively. The alloy bit was in general more satisfactory than straight carbon steel. However, the loss of bits in the mines, coupled with the possibility of using scrap cruciform drill steel, offset the economic advantages of the alloy steel; therefore, 1½-in. light section cruciform having a carbon content of 0.80 to 0.90 per cent., and manganese 0.20 to 0.30 per cent., was adopted as a standard for bit steel. The bars are received from the mill in 18 to 20-ft. lengths, and after being heated to a temperature of 400° F. in an open flame are cut in a jaw shear into slugs 1⅝ in. long and weighing 10 oz., for starter or 1⅞-in. bits. The slugs are placed in a hopper above the forging furnace so that the escaping gas preheats the work before it is raised to the forging heat of 1800° to 1900° F. The screw press forms the bit and a trimmer removes about 1 oz. of flash and drops the forging into a cast-iron container 8 in. dia. and 24 in. high, holding 250 starter bits each. At intervals charcoal is added, and the full container is completely covered with a layer of charcoal to prevent oxidation during the annealing period. After cooling to room temperature, five such containers are charged into an oil-fired muffle furnace. Four hours are required to reach the annealing temperature of 1450° F.; the forgings are held at this temperature for a further period of 4 hr. and then slowly cooled to 700° F. before the furnace is opened. This gradual heating and slow cooling produces a spheroidized structure providing maximum machinability as a Rockwell hardness of 90 B. scale indicates.

Because air and oil are used to cool and lubricate the forging dies, a small amount of Fe_3O_4 forms on the forged bit. Consequently the annealed bits are cleaned of all scale in a rotating tumbler before they are sent to the milling machine.

The bits are milled by a 12-in. automatic vertical miller equipped with special jigs and fixtures for this work. Four bits are clamped on an inclined table and rapidly advanced to the cutters, where the table feed is reduced to 2½ in. per minute. One-half the wing bevels are milled on the first pass. The table is rapidly returned, indexed by hand through 180°, and the operation repeated to finish the milled bit. A gang of 10 cutters is mounted on the machine arbor, and about 3000 bits can be milled before it becomes necessary to grind the cutters. The milled bits are gaged at frequent intervals and errors in cutter adjustment or table height are detected and corrected before further production is permitted.

In order to remove surface defects and decarbonization and to provide a clean, smooth surface for hardening, the milled bits are ground on a Maickel automatic grinding machine. A similar machine, but using a different grade of wheel, is used to grind the dull bits received from the mines. The grinding machines were developed and patented by Joseph

Maickel, an employee of the Hawkesworth company, and their introduction increased grinding operations from 700 to 1400 bits per man per day.

The equipment consists essentially of a motor-driven shaft, revolving at 1160 r.p.m., upon which are mounted five grinding wheels, four for facing and one for edging. The wheels are 20 in. dia., 1½ in. thick, and 8-in. bore. The facing wheels are beveled to give an included angle at

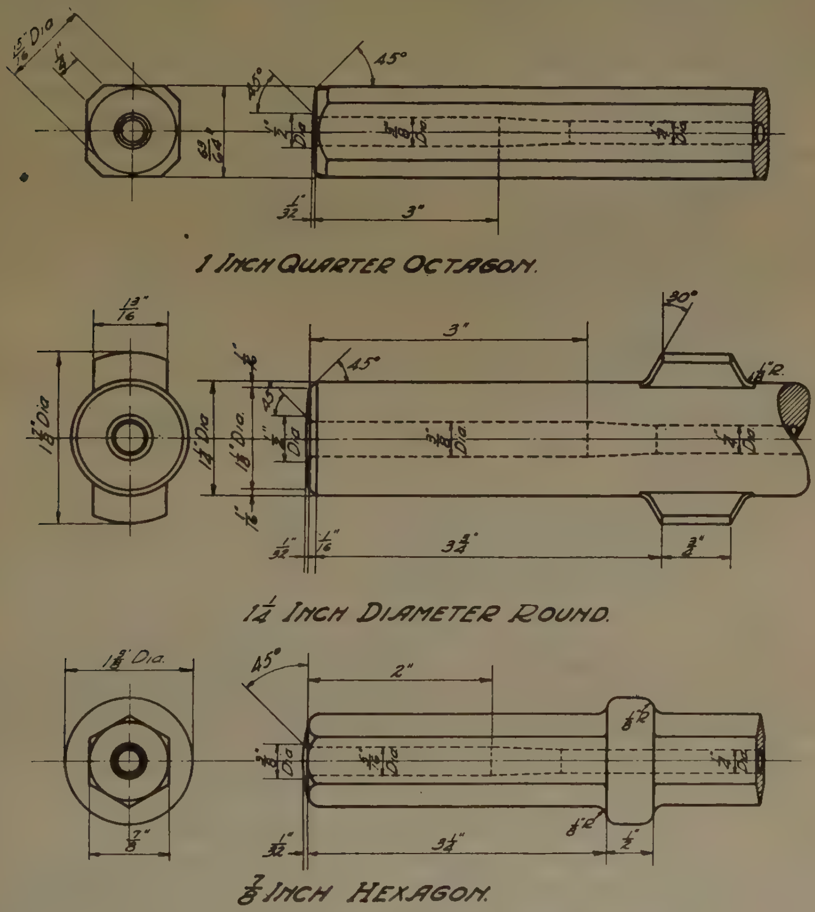


FIG. 3.—MACHINE ENDS OF HOLLOW DRILL SHANKS.

the intersection of 120°. The edging wheels are straight faced. Diamond dressers attached to the machine frame are used to surface the wheels. The work is held in two revolving, adjustable drums placed in front of and in line with the grinding wheel. The facing drum holds 40 bits spaced in four rows of 10 each. The bits are automatically turned 90° each revolution, thus bringing each face in contact with the grinding wheel. The edging or gaging drum mounts 22 bits, which are automati-

cally turned to give contact with the grinding wheel through the entire circumference. Both the regrounds and new bits coming from the grinding machines are inspected before hardening. Those passed by the inspector are placed on steel trays holding 120 to 150 bits, depending on the size, and are then ready for the final heat treatment.

Heating for hardening is accomplished in electrically heated lead and salt baths. The alloy pots for these baths are 14 in. dia. and 18 in. deep and require a maximum electrical input of 22 kw. each. Two thermocouples, one located near the heating element and one in the bath itself, automatically control the temperature. The bits, while heating, rest on perforated steel plates, immersed to the desired depth in the respective baths.

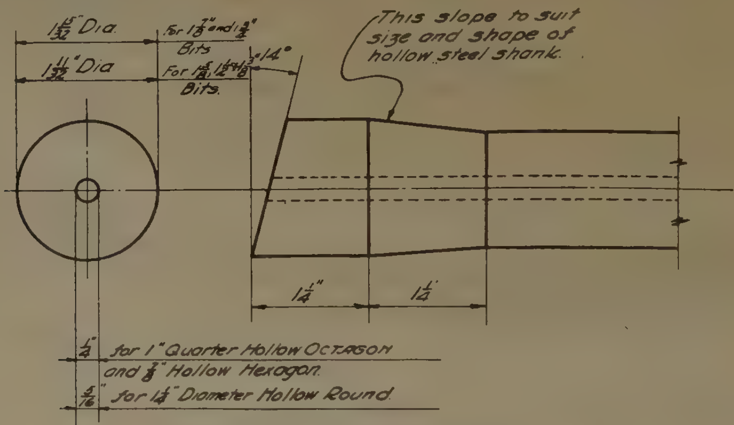


FIG. 4.—FORGED BIT END OF HOLLOW DRILL SHANKS.

Adjacent to the salt bath is a water quenching tank, 23 in. wide, 38 in. long, and 12 in. deep. A steel manifold 12 in. wide, 26 in. long, 1 in. deep is suspended in this tank. Intake and discharge water connections are placed at opposite ends of the manifold, through which water is continually circulated. The top of the manifold is perforated with 12 groups of holes spaced at 4-in. centers. Each group consists of five $\frac{1}{16}$-in. dia. holes and are arranged so that the cold water forced through under a slight pressure strikes the bit at each of the four corners and the center. The water level can be adjusted so that the bit can be immersed to approximately one-third of its height.

The hardening operation begins with the placing of 24 bits in the lead bath, which is maintained at a temperature of 1000° F. After a heating period of three minutes, this work is transferred to the salt bath, at a temperature of 1450° F., and held therein for a further period of three minutes. The bits are then individually withdrawn and placed on the quenching manifold described above. When the color in the tongue disappears they are immediately removed and completely immersed in a

tank of boiling water, where they remain for a period of 3 to 4 hr. The heat treatment produces an unwarped bit, decreasing progressively from a 65 Rockwell hardness of the cutting edge to a comparatively soft tongue, and wing structure. After a final inspection, the product is sent to the shipping department for distribution. The dull bits received at the plant are routed directly to the Maickel grinders. With the exception of the grade of wheel used, these machines are identical with those used for grinding new bits. A dull starter or 1⅞-in. dia. bit is

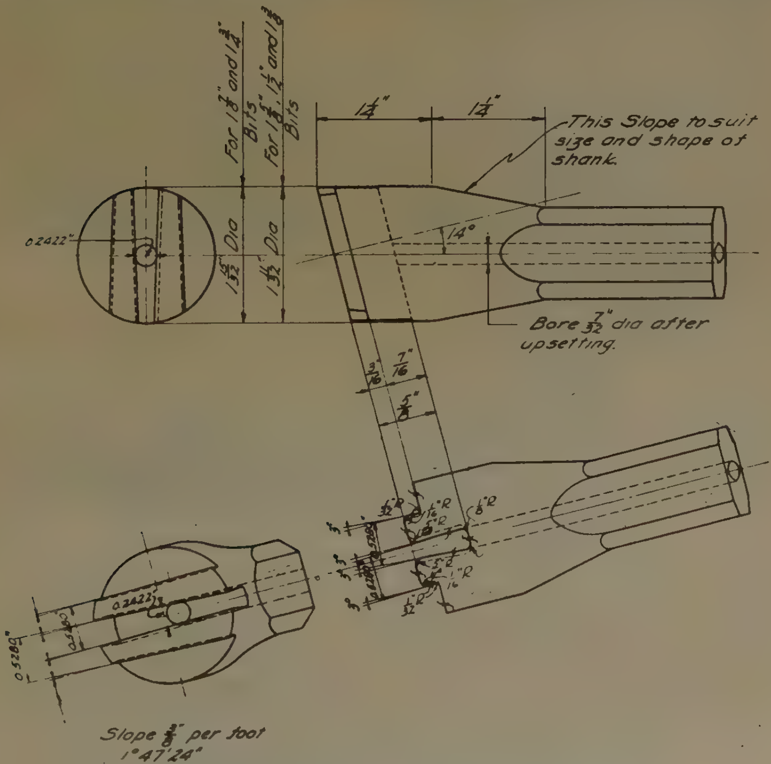


FIG. 5.—MILLED BIT END OF HOLLOW DRILL SHANKS.

ground to a second or 1¾-in. size. Butte practice requires that the bits be successively ground to a fifth of 1⅜-in. dia., after which they are discarded. The grinding operation draws the temper so that all reground bits are given the same heat treatment accorded the new product. The ground bits are carefully inspected. An average of 2 to 3 per cent. fail to pass the inspectors and are returned to the machines for further grinding.

Most of the shanks are made of 1-in. quarter octagon hollow drill steel having a carbon content of 0.75 to 0.85 per cent., and manganese 0.25 to 0.35 per cent. The steel is received from the mills in 18 to 20-ft. lengths and cut to required lengths by a power hacksaw. An additional

length of 1¾ in. is allowed for upsetting and forming the bit end of the shank. The steel is heated to 1850° F. in an open oil forge, upset on a drill sharpener and permitted to cool to room temperature.

The shanks are then transferred to an oil-fired annealing furnace and the temperature raised to 1450° F., and slowly cooled to room temperature in a lime box. Surface scale is removed by hand grinding before milling, which is accomplished by a 12-in. vertical miller equipped with

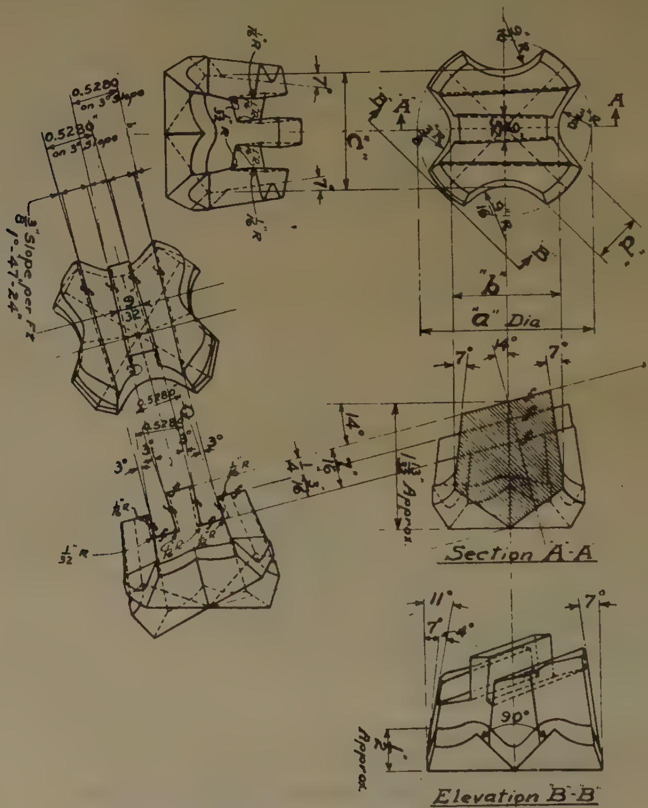


Fig. 6.—Hawkesworth detachable bits.

special fixtures. The milled shanks are racked on trucks and delivered to the hardening furnaces. The temperature of this furnace is so controlled that the cold steel is slowly raised to a maximum of 1100° F. and then quickly raised to the hardening temperature of 1450° F. The shanks are quenched in oil and then drawn to a temperature of 575° F. for a period of 15 min. The chuck end of the shank receives the same preheating, but the hardening temperature is raised to 1650° F., and this end is not drawn after quenching in oil. The procedure for damaged shanks is practically the same as that for new steel.

The plant acquired from the Hawkesworth Drill Co. has not the capacity to supply all the Anaconda company property located in Butte. To meet this demand a new plant is now in the course of construction.

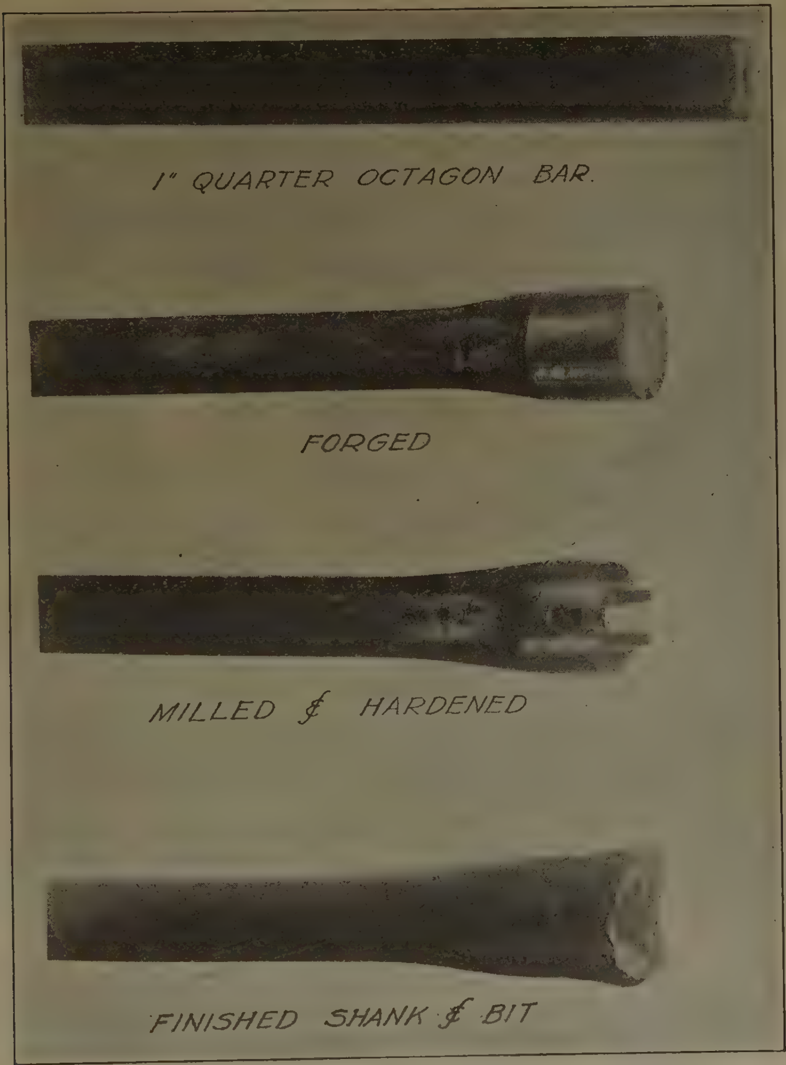


Fig. 7.—Steps in manufacturing shank and bit.

Equipment will be installed to produce 3000 new bits, 4000 reground bits, and 800 shanks per 24-hr. day.

Fig. 2 shows the general layout of the equipment, which will be housed in a brick building having a floor area of approximately 10,500 sq. ft. The manufacturing scheme will be essentially the same as that heretofore

described. However, new equipment will be installed, which will reduce the labor required for operation and materially increase the production per unit. A combined shear and forging machine will supplant the present jaw shear and screw press. This machine has a capacity of 3600 bits per hour, providing steel can be heated and fed at the maximum speed. Forging flash will be reduced 50 per cent. from present practice. Similar equipment will be installed to upset both the chuck and bit end of the shanks. Two electric bit-annealing furnaces having double the capacity of the present oil-fired furnace will be included in the new equipment. Electric annealing permits the discarding of the cast iron containers, with their charcoal seals, since very little oxidation occurs in this type of furnace.

Specially designed bit and shank millers, having 40 per cent. more capacity than the present machines, will materially reduce the cost of this operation. An Oxweld cutting machine will replace the power hacksaw for cutting new drill shanks. Refinements have been suggested for the Maickel grinders and will be incorporated in all new machines.

The remaining equipment is similar to that used in the present establishment except that, in most instances, the units are doubled.

The accompanying drawings (Figs. 3 to 8) show bit and shank dimensions, and the photographs indicate the various steps in the manufacturing process.

Research work related to the improvement that might be possible if other compositions in steel were used for both shank and bit is being conducted at the present time.

Manufacturing methods are satisfactory and costs will be reduced by 20 per cent. in the new plant.

Comparison of Regular Steel and Detachable Bit

The first comparison of interest is the comparison using the all mines regular drill steel costs in 1927 and the cost of detachable bit steel at the Badger State mine for the first six months of 1929, both of which, as stated before, were obtained by a special detailed study.

Regular Drill Steel

All mines regular steel cost in 1927 was $0.003618 per cu. ft. excavated.

Badger State mine excavation, first six months, 1929, was 3,257,496 cu. ft.

Applying the all mines cost of $0.003618 per cu. ft. to the Badger State excavation for six months of 1929, we have 3,257,496 cu. ft. × $0.003618	$11,785.62
Cost of steel distribution by toolmen for six months	4,133.60
Cost of topmen, station tenders and miscellaneous distribution	1,803.95
Cost of distribution by miners' time	8,143.74
Total regular steel cost, six months	$25,866.91

Detachable Bit Steel

Average Badger State mine 1928–1929 tests = $0.005052 per cu. ft. excavation.	
Badger State mine excavation six months, 1929 = 3,257,496 cu. ft., 3,257,496 × $0.005052	$16,456.86
Distribution for detachable-bit steel	826.72
Total cost	$17,283.58
Net saving = $8,583.33, or $17,166.66 per year.	

The next comparison is made between the average regular drill steel cost of 1927 and the average all mines detachable-bit costs of 1928–1929,

FIG. 8.—STEPS IN MANUFACTURING BIT.

applied to the Badger State cubic foot excavation for the first six months of 1929.

Regular Drill Steel

Total regular steel cost, including distribution and miners' savings as shown before, is $25,866.91.

Detachable Bit Steel

Average all mines 1928–1929 detachable cost per cubic foot excavated = $0.004852.	
Badger State, first six months, 1929, excavation cost	$15,805.37
Toolmen distribution	826.72
Total detachable cost	$16,632.09
Net saving for six months	9,234.82
Net saving for one year	18,469.64

All Mines Comparison between Regular and Detachable Bits on the Basis of Cost per Cubic Foot Excavated

	Cost Cu. Ft. Excavated
Regular drill steel	$0.003618
Toolmen in mine	0.001237
(Average of all mines 1927, 1928, and 6 months, 1929)	
Station tenders, topmen, and engineers—basis, 1927	0.000553
Cost of distribution by miners	0.002500
Total cost regular steel	$0.007908
Detachable bit steel	0.004852
Toolmen in mine	0.000247
Total cost detachable bit steel	$0.005099
Total net saving per cubic foot excavated in favor of detachable bit steel	$0.002809

When this saving at all mines is applied to the Badger State mine cubic foot excavation the saving is $18,300.60 per year, which checks closely with the results of the Badger State mine test figured by its own independent costs, which, as shown above, was $17,166.66 and $18,469.64 per year.

Conclusion

All Mines Detachable Bit Steel Record in 1928 *and* 1929

In order to show the volume of use and work done with detachable bits in arriving at a final conclusion to standardize our drilling operations by the use of Hawkesworth detachable bits, there is presented below the totals of the various items which were used. The total all mines cubic foot excavation with all drill steel during the year 1928 and the first six months of 1929, was 80,782,661 cu. ft., while the excavation credited to Hawkesworth drill bits during that period was 28,228,664 cu. ft., or approximately 35 per cent. of the total excavation was made with detachable bits.

Statistics for the Period

New bits used	432,040
Reground bits used	712,941
Total bits used	1,144,981
New shanks used	14,794
Repaired shanks	66,586
Installation shanks	7,573
Total steel supply money	$136,969.58
Total cubic foot excavation—detachable	28,228,664
Cost per cubic foot excavated, less installation	$0.004852
Total saving on 28,228,664 cu. ft., using $0.002809 per cubic foot, which was the saving over regular steel.	$79,284.31

Considering 50,000,000 cu. ft. as the average total annual cubic foot excavation of ore and waste, the Anaconda company will save $140,450

per year by the use of Hawkesworth detachable bits, based upon the above figures.

However, with reduced manufacturing costs, improvements in distribution, control of loss in mines and other factors, we feel that this saving will be materially increased when production comes from an ideal plant installation, and with the service established on a positive operating system, not obtainable through the test period.

Throughout the test period of 1928 and 1929 the excavation per bit used was 24.5 cu. ft. per bit, but under conditions at the present time this average has been 30 cu. ft. per bit, or approximately that of regular drill steel.

The positive savings are reflected in the various operating cost factors as follows:

A more uniform bit and shank than is possible with regular steel. This applies to details of temper, gage, and cost.

A control plant rather than a drill steel shop at each mine.

Time saved in the distribution cycle through shafts to working place and return. This amounts to several hours per day per mine on available hoisting hours of shaft time and hoisting equipment. It is, perhaps, the most important item at mines where maximum production is maintained. The same would apply to tunnel or subway operations.

The saving in time to all miners when ease of transportation to working places and adequate drill steel supply is considered.

The safety factor is most important in shafts, raise work, and general distribution facilities.

The mining department staff of the Anaconda Copper Mining Co. standardized on this equipment after a long detailed study and application of it to all operating requirements, and are satisfied that it is a safer, more efficient, and cheaper tool than regular drill steel.

The problem of supply is most important. In this case the situation was ideal in lending itself to possible advantages. This should be true in other large operations.

As stated before, the Anaconda Copper Mining Co. owns its own plant and is independent of the Hawkesworth Detachable Drill Co. The Anaconda company is manufacturing the shanks and bit under special arrangements for use in its Butte mines, under the direct supervision of its own mechanical engineering department.

The Hawkesworth Detachable Drill Co., of Butte, Mont., is forming plans at the present time for the production of shanks and bits in various parts of the United States. It has been awaiting the final conclusion of the Anaconda company.

The publication of this paper or any other record on this subject has been withheld up to the present time because of the desire to await the time when a positive statement could be made concerning the merits of this equipment under all conditions of service, efficiency, and costs.

DISCUSSION

C. M. Haight, Franklin, N. J. (written discussion).—A connection between the bit and shank pieces, which will stand up to the drilling service and yet which can be separated easily and quickly, has been the problem in the development of detachable bits for rock drilling. The connection has also to be kept less in diameter than the smallest gage of bit used. This might at first thought seem a problem of easy solution; but much experimentation, with high hopes often dashed to pieces, has been the lot of those who have attacked the problem. Now, however, practical solutions have been achieved.

Besides the Hawkesworth type of joint, which uses a sort of dovetail arrangement, much experimental work has been done with a connection using a threaded coupling to connect a shank piece with a threaded end to a short bit piece, similarly threaded. This type has been developed so that it is now being manufactured commercially, for use by contractors, with success and satisfaction.

At present these bits are discarded when dulled, since the users are scattered over such a wide territory that return costs make the throw-away policy advisable. Whether such a policy will be continued will depend largely on future developments in the use of these bits. A similar question may arise should the Hawkesworth bit be made for sale to mines and construction jobs.

Much study will have to be given to the matter of actual cost of the drill-steel methods now in use by the individual operations, to determine the conditions under which the use of detachable bits will be advantageous.

Mr. Berrien's figures show that the expenditures made in experimenting with and developing the Hawkesworth bit have been amply justified by the savings now possible through the use of these bits. Since those engaged in drilling rock now have these types of bits presented to them in practical form, the extent to which the use will spread (with a resulting research and improvement) is distinctly up to those who use drill steel. New projects, where no sharpening equipment will have to be discarded, should give this type of equipment especial consideration; and other projects should approach it without prejudice.

The amount of follow-up described by Mr. Berrien seems, at first thought, more extensive than is necessary; but where parts as small as these bits are used the tendency for loss or even discard will be high unless some check is used. The method developed at Anaconda may well serve as a guide until experience shows a variation advisable. A close check on parts will be wise when first beginning to use this equipment. In fact, a greater follow-up of regular drill steel than is generally used might be worth while.

E. V. Daveler, Butte, Mont. (written discussion).—In our work here, at the Butte and Superior Mining Co., we have, for the past 12 years, paid considerable time and attention to the subject of drill steel and its treatment. Comparisons of the life of drill steel up until 1925 showed that the treatment and use was about all that could be expected and that further improvement must either be in the steel itself or in the adoption of some mechanical improvement along these lines.

We had noted with interest and kept in touch with the experimental work and the early development of the Hawkesworth detachable bit and were also familiar in a general way with the experimental work carried on by the Anaconda Copper Mining Co. With the Hawkesworth company on a production basis in November, 1924, the Butte and Superior company began testing the Hawkesworth bit early in 1925. The first work was on a limited basis in May, 1925. At the same time the Badger mine, of the Anaconda company, was being equipped throughout for a test run with the Hawkesworth bit. Early in July, 1925, two of our levels were completely equipped

and in September, two additional levels were equipped throughout. Early in the year 1926 the entire mine was equipped with Hawkesworth bits and operations have been continued with the Hawkesworth bit since that time—the Butte and Superior Mining Co. being the first company to adopt the use of the bit throughout its entire operation.

During the two years 1928 and 1929, operations have been as follows: 10,185,201 cu. ft. were broken, for which 93,198 new bits were used and 164,332 bits used for regrinding, a total of 257,530. The total loss and use of shanks for the two-year period was 40,357 lb.; the total cost for the period was $56,598.40, and the cost per cubic foot excavated was $0.00555. The cost of regular steel, considering supplies and labor necessary in sharpening, was $0.00396 per cubic foot, to which, however, must be added the additional labor involved in the use of the regular steel for nipping. This varies greatly in different properties, but a minimum net expense for this account at the Butte and Superior property was about $1040 per month or $0.00244 per cubic foot, or a total cost using regular steel of $0.00640, which compares with $0.00555.

The saving estimated under the Hawkesworth as shown above does not consider savings which we term intangible, such as time and power saved in hoisting, time saved by the men themselves in not having to handle the long steel, surface handling expense, etc. This saving of time by the men actually using the steel is, of course, one of the most important savings involved by the use of the detachable bit, and in Mr. Berrien's review, time studies have furnished estimates of this saving.

The matter of the use of the Hawkesworth has been covered so thoroughly by Mr. Berrien that there is not a great deal to add. Our operations vary somewhat from those described by Mr. Berrien. They have standardized on 1-in. quarter-octagon steel throughout whereas we are using 1¼-in. round steel for all drifting and 1-in. quarter octagon for all stoping. On knock-off blocks we find that cast steel is cheaper in the long run than cast iron. The bit loss varies from 6 to 10 per cent. of the bits dulled and from 1 to 2 per cent. of the bits issued to the miners.

In considering the adoption of Hawkesworth bits, a thorough record should first be kept in complete detail of the costs of steel in use, using the regular steel over a sufficient period of time so that the underground loss of steel can be carefully calculated. Such a record was kept at the Butte and Superior Mining Co. over an 18-month period, with an inventory at the beginning and end of this period; at the same time a record of cubic feet broken, segregated into slopes, raises and sills, should be kept, as in afterwards making a comparison with the Hawkesworth, an increase or decrease in development work in proportion to the tonnage broken in the stopes might vitiate the comparison made.

During the past few years the method of manufacturing the bits has been steadily improved and it can be expected that this improvement will continue, with corresponding reductions in manufacturing costs and cost to the consumer, so that the ultimate saving through the use of the bit will be still greater at properties where its economical use has already been demonstrated.

Observation on Ground Movement and Subsidences at Rio Tinto Mines, Spain

By Robert E. Palmer,* London, England

(New York Meeting, February, 1930)

So much has already been written on this vast subject of ground movement and subsidence, and so many data collected and commented upon, that in this paper the author proposes to confine himself to the submission of some plans showing what has actually occurred in the mining of several of the orebodies which have come under his direction during the last 20 years or so at Rio Tinto, Spain. Some notes are added for the purpose of describing the plans. The author does not attempt to give his opinion as to the reasons why the subsidences have occurred in the manner in which they are found, but confines himself to giving as true a picture as possible of the movements that have taken place.

Effects of Excavation on Ground Movement

Broadly, the effects of excavations on the overhead or adjacent ground can be classified under three headings, as follows:

1. Ground movements due to excavations made from surface only, such as those made in opencasts where no excavations underlie them. The stability and strength of the ground to resist movement must determine the slopes.

2. Ground movements due to underground excavations only; *i. e.*, where the surface is not removed.

3. Ground movements due to a combination of both; *i. e.*, from excavations taking place from the surface for the purpose of stripping an orebody and recovery of the ore to a certain horizon combined with excavations being carried out below, at the same time, for the winning of the ore below the horizon at which mining by stripping or opencasting under the specific conditions is no longer economical.

Nos. 1 and 2 represent independent and distinct problems and No. 3 is a combination, which presents results much more difficult of explanation. The accompanying plans and sections give examples of what has occurred under the three conditions. Unfortunately, the class of ground is not exactly the same in them all, but the difference is not greater than is generally encountered over districts of equal area.

Excavations from Surface

To illustrate No. 1 three cross-sections are shown, of excavations made for the stripping and excavation from three different orebodies

* Consulting Mining Engineer.

(Figs. 1, 2, 3). Fig. 1 is a section through the South lode on line 570 as shown on Fig. 16. Only the right-hand or porphyry side need be referred to for the moment, as the left-hand or slate side will be dealt with later on.

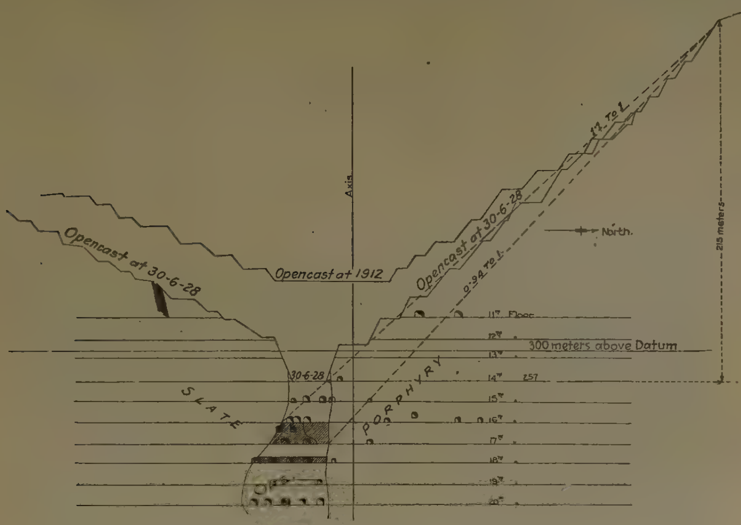


FIG. 1.—SOUTH LODE. CROSS-SECTION ON LINE 570 OF FIG. 16.

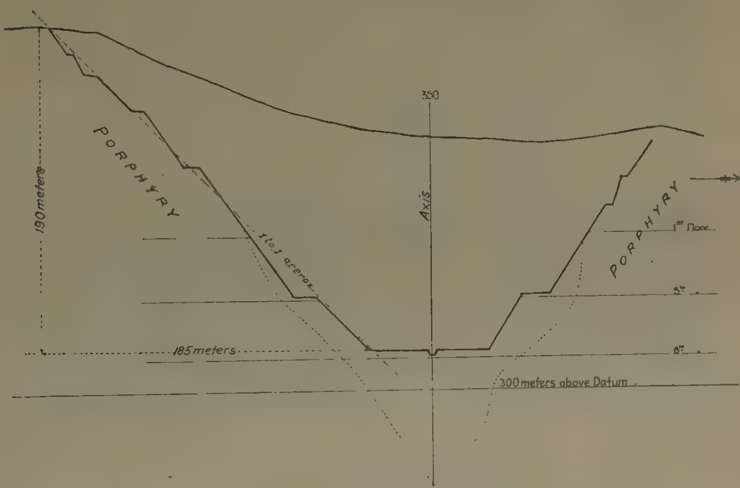


FIG. 2.—NORTH LODE OPENCAST. CROSS-SECTION SHOWING BANK STOPES EXCAVATED BY HAND.

The porphyry is a highly siliceous rock, somewhat decomposed and inclined to schistosity in the direction of the axis of the orebody. The vertical depth of the cut at present is some 215 m., while the horizontal component is 237 m., giving a slope of about 1.1 to 1. This may be considered as the steepest angle at which this rock will stand, excepting in

isolated zones, and apparently it will stand at this slope more or less permanently. It is not subject to the action of snows and frosts. An underground excavation, about 36 m. wide and 12.5 m. high, has been made and filled at some 30 m. below the toe of the slope and, to date at all events, no further movement at this section is noticeable. A line drawn from the bottom of this excavation to the top of the slope gives a ratio of 0.94 to 1, and it is fairly certain that had no stripping taken place the ground movement would have been much steeper than this, so that at least until excavation takes place at a lower horizon the benches will remain intact.

Fig. 2 shows a section taken through the North lode opencast, an outlying orebody of which no plan is shown. Here the orebody lies entirely enclosed in the same type of rock as that shown on the right-hand side of Fig. 1; that is, siliceous porphyry. The total vertical height is 190 m. and

FIG. 3.—SAN DIONISIO OPENCAST. CROSS-SECTION ON LINE 2050 OF FIG. 5.

the horizontal distance about 185 m., giving a slope of just about 1 to 1, and that appears to be the limit at which it will stand for this depth.

Fig. 3 shows a section taken through the San Dionisio opencast on line 2050 of Fig. 5. Fig. 4 is a front view of the right-hand or porphyry slope. The total vertical depth on this side to date is 170 m. and the prevailing slope about 1 to 1. Stripping is being continued, but although the porphyry here is very tough and homogeneous, it is doubtful if it will stand safely at a much steeper slope than that shown on the upper portion; *i. e.*, about 1 to 1.

UNDERGROUND EXCAVATIONS

To illustrate No. 2 are submitted a plan (Fig. 5) and two sections (Figs. 6 and 7), together with five photographs (Figs. 8 to 12). The positions from which and the directions in which the photographs were taken are shown on Fig. 5, the plan of the San Dionisio lode opencast and the surface east of it which overlies the underground workings on the lode. The photographs were taken to show the cracks and surface subsidence caused by underground excavation.

In this case we have an orebody lying—so far as the part under consideration is concerned—on the contact between a highly siliceous porphyry dike on the north, forming the footwall of the lode, and a fairly tight and true slate on the south, or hanging wall. The material overlying the orebody consists of loose debris, the result of decomposition of the igneous and sedimentary rocks and of the residues from the decomposition of the ores. The porphyry is hard and tough and, as a general rule, there is a

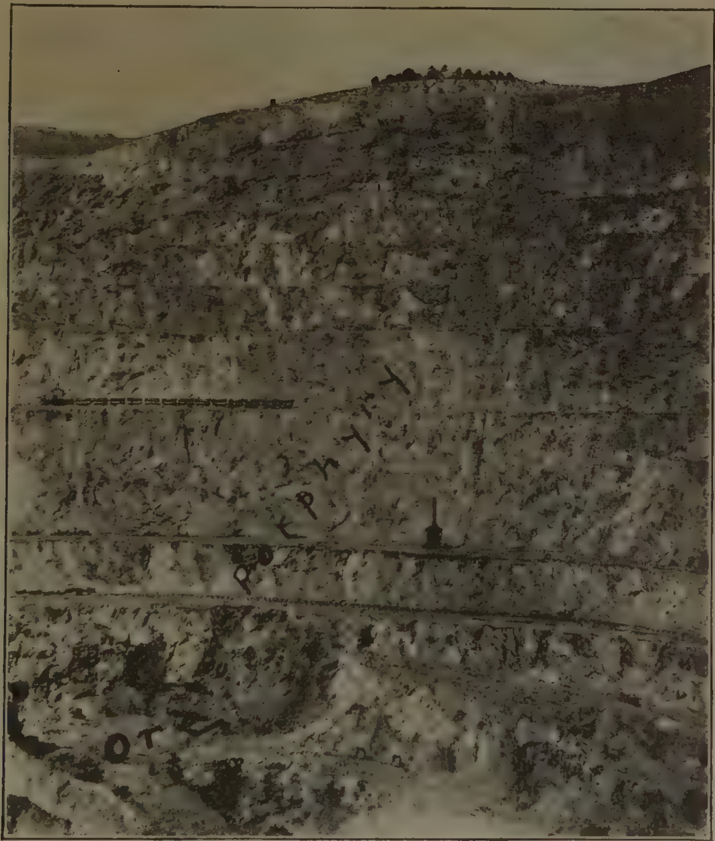


FIG. 4.—FRONT VIEW OF PORPHYRY SLOPE, SAN DIONISIO LODE.

clean line of contact between it and the ore, which consists of iron and cuprous pyrites. The ore is frozen to the wall rock.

Figs. 8, 9 and 10 are views of the main crack along the south or slate side of the orebody, which appeared in 1920. Figs. 11 and 12 are general views taken from points farther east.

Originally this orebody was mined by a system of stall and pillars on floors 12.5 m. apart, but the portion removed was so small that no ground movement was observed. It was only after the system was changed to

that of cut and fill, and when all the ore was removed from the excavation, that surface disturbance began. The excavation is carried out by the top slice and fill system; *i. e.*, horizontal slices some 2.5 m. thick are taken off at the top of the existing ore and the resulting spaces are filled. This

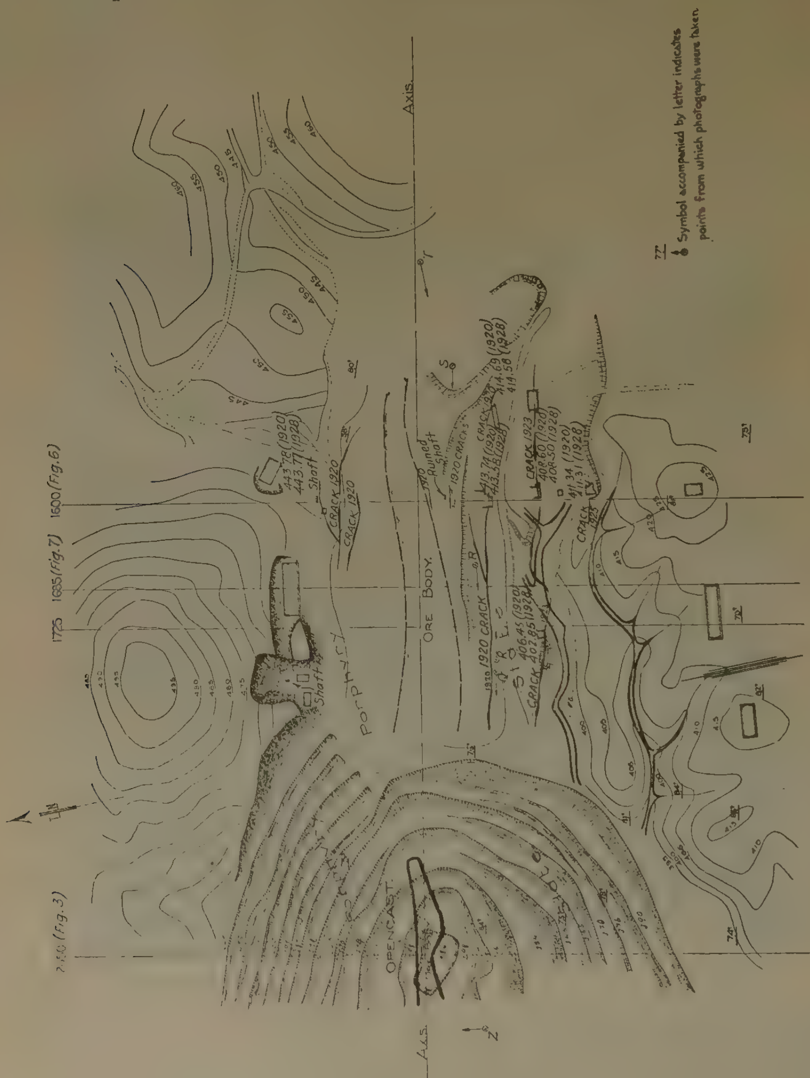


FIG. 5.—PLAN OF SURFACE OF SAN DIONISIO LODE.

is carried out from three original floors simultaneously, but so far as subsidence is concerned the lower or third floor cuts may be left out of the picture, as they are not sufficiently far advanced to take the weight until the two floors above have been finished; that is, in general only two floors, possibly 25 m. thick, are being removed at the same time.

The filling is placed as the excavations are made. It consists of stone, generally broken porphyry being excavated in the stripping of other portions of the lode or other lodes in sizes that the ordinary laborer can handle, with the corresponding quantity of fines. It is packed as tightly as practicable under the circumstances, but naturally, before the weight

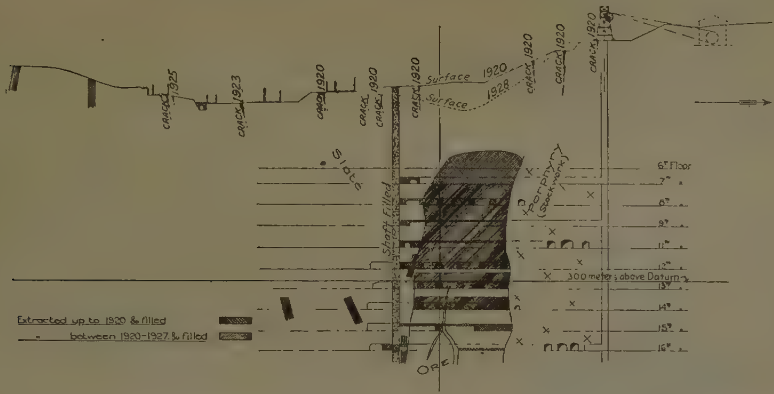


FIG. 6.—SAN DIONISIO LODE. CROSS-SECTION ON LINE 1600 OF FIG. 5.

comes on it, there are many voids. Tests run on the same material packed in a similar manner go to show that the filling as hand-packed contains some 25 to 30 per cent. voids; that is, if it were possible to replace the excavation by a solid piece of rock, the latter would fill only from 70

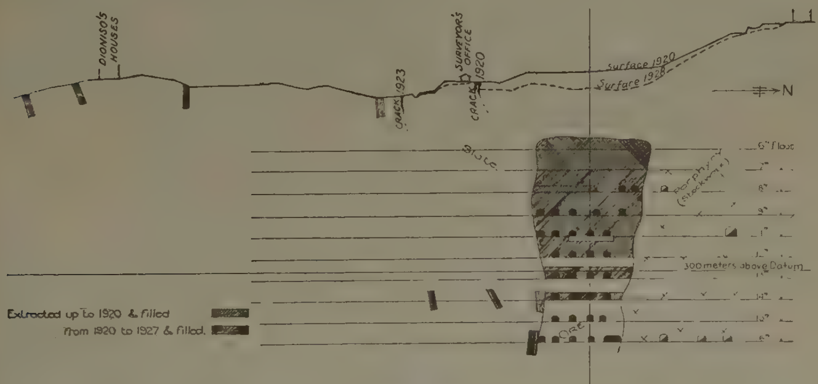


FIG. 7.—SAN DIONISIO LODE. CROSS-SECTION ON LINE 1685 OF FIG. 5.

to 75 per cent. of the space, allowing the remaining space for settlement of the surface.

When the portion above the seventh floor, as shown in the two sections (Fig. 6 and 7), had been removed and filled, the surface began to settle and the cracks marked 1920 became visible. The sections are

taken across the San Dionisio lode in the lines 1600 and 1685, respectively, of Fig. 5. They show the position and extent of the underground workings and the positions of the cracks which develop at different times as a result of these workings. The presumed northerly boundary of movement may be judged from the location of the excavation and the crack that is farthest away.

As mentioned, the south or hanging wall consists of a fairly true slate having its bedding or parting planes fairly vertical, though with a tendency to dip to the north. The strike of these bedding planes is more or less parallel with the orebody.

FIG. 8.—MAIN CRACK ALONG SLATE SIDE OF OREBODY, SAN DIONISIO LODE.

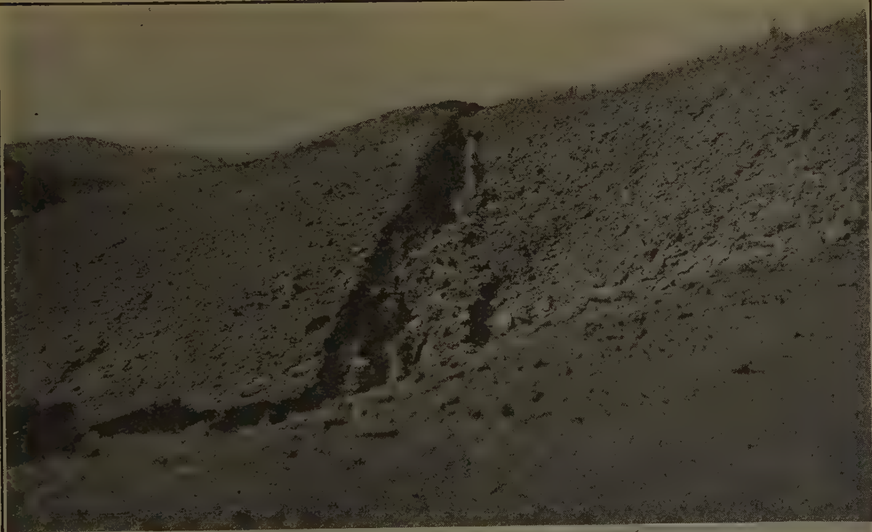
The sections show the major cracks in this slate and the date upon which they were first seen; none of them appeared until about the date upon which all the ore above the seventh floor had been mined and filled. As seen on the surface, all these cracks appear as if they continued down in a vertical direction, cutting across the bedding planes of the formation. How far they continue vertically before changing direction is not known.

The subsidence of the surface during the period between the years 1920 and 1928 is shown on all the sections, and the corresponding excavation during the same period is also indicated.

Surface and Underground Excavation

Under heading No. 3 are shown some of the results of carrying out the two methods of mining simultaneously; that is: opencast mining to a depth below which the cost of the ore won would exceed the cost of mining

by underground methods together with underground mining of the portion where the stripping and excavation of the ore would exceed the cost by underground methods.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10

Fig. 9.—Looking west on crack from point about 100 m. south of San Dionisio shaft.
Fig. 10.—Looking east from point about 100 m. south of old main shaft.

Six sections and a plan are submitted (Figs. 13 to 19). The section in Fig. 13 is taken across the South lode on line 420 shown on Fig. 16. Here again, the porphyry dyke appears on the right-hand or north side and the slate on the left hand or south side.

The rocks are more or less the same as mentioned under No.2, excepting that in this case the porphyry is more decomposed and leached and less homogeneous and tough, while the so-called slate on the south side

11

12

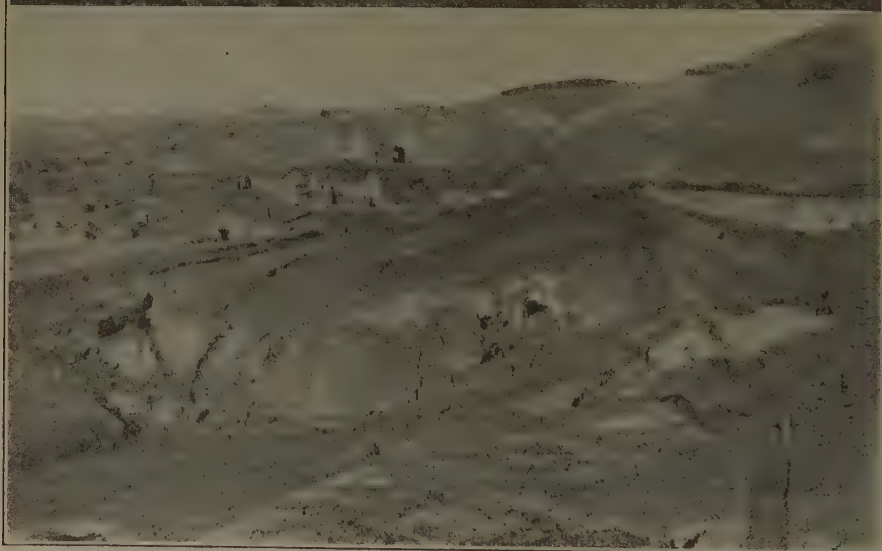

Fig. 11.—Looking west from east end of subsidence over orebody.
Fig. 12.—Looking west from "Old" Edwards shaft.

is softer and less baked. In fact, it has been termed an indurated mud. It has, however, very distinct and well defined bedding planes and, up to the date when movement began, wells sunk in it for the purpose

of catching surface waters were tight, and little percolation, if any, took place.

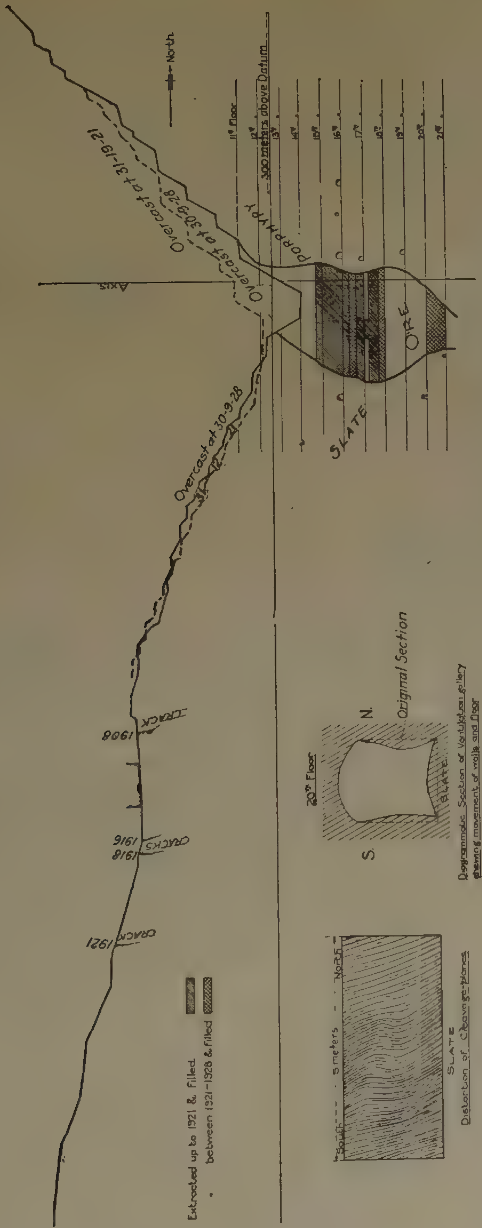

FIG. 13.—SOUTH LODE. CROSS-SECTION ON LINE 420 OF FIG. 16.

The bedding planes have a strike almost parallel with the axis of the lode and a dip to the north or right hand.

This orebody (South lode) was extensively mined by pillar and stall, so much so that ground movements took place before the real excavation by cut and fill began, and in this particular section (normal to the axis of the orebody) cracks appeared as shown from 1908 to 1918 with slight movements underground from about the sixteenth floor upwards.

During the period 1921–1928, both inclusive, no stripping has taken place on this south side, although the excavation has been made both underground and from the toe of the original stripping. The result of these excavations, either combined or separately, has been that the ground has settled near the top of the slope and risen on the lower half. This is shown in greater detail in Fig. 14 (the same section as in Fig. 13, on a larger scale), where it will be noted that the slates have tipped over and been pushed up, the present position being higher, not lower, than in 1921. The angle at which they lie is truly given on the section.

An interesting point is the appearance of the gallery on the twentieth floor. This was a return airway until it had to be abandoned and replaced recently. This gallery was supported by a masonry arch. As it began to collapse, it was supported by ordinary timber sets. Apparently, there is no weight on the top, all the pressure coming from the sides. The surface cracks are more or less vertical to an unknown depth, the pressure below is horizontal and the question is, where is the line or curve of fracture?

Fig. 15 shows another section of the excavation made on the same lode, taken across line 360 on Fig. 16. In 1920, a series of cracks opened on the porphyry side. These were traceable, as shown on the surface, at the top of an underground shaft on the eleventh floor, in a crosscut on the thirteenth floor and, although not shown, were perceptible in the wall rock on the sixteenth floor. This occurred during cut and fill excavation of the block between the sixteenth and seventeenth floors. Surface examination led to the impression that this movement had taken place along a sheared zone on the porphyry. This impression, and observation of the section of the stripping at that date, led to the supposition that this movement was entirely due to underground excavation and not to inadequate slopes.

On the slate side, in the year 1908, two well-defined cracks opened on the surface a long distance back from the cut. These were not due to full excavation by cut and fill method, because at that date this method had not been commenced. They were caused by a crushing together of some of the old stalls reaching as far down as the eighteenth floor. No records are available as to what actually happened on the benches of the opencast shown as at 31.12.1907, but as all the benches disappeared and were converted into a slide, it is to be presumed that the same thing happened as that shown in Fig. 13; *viz.*, that the portion between the cracks dropped apparently vertically, raising and breaking up the

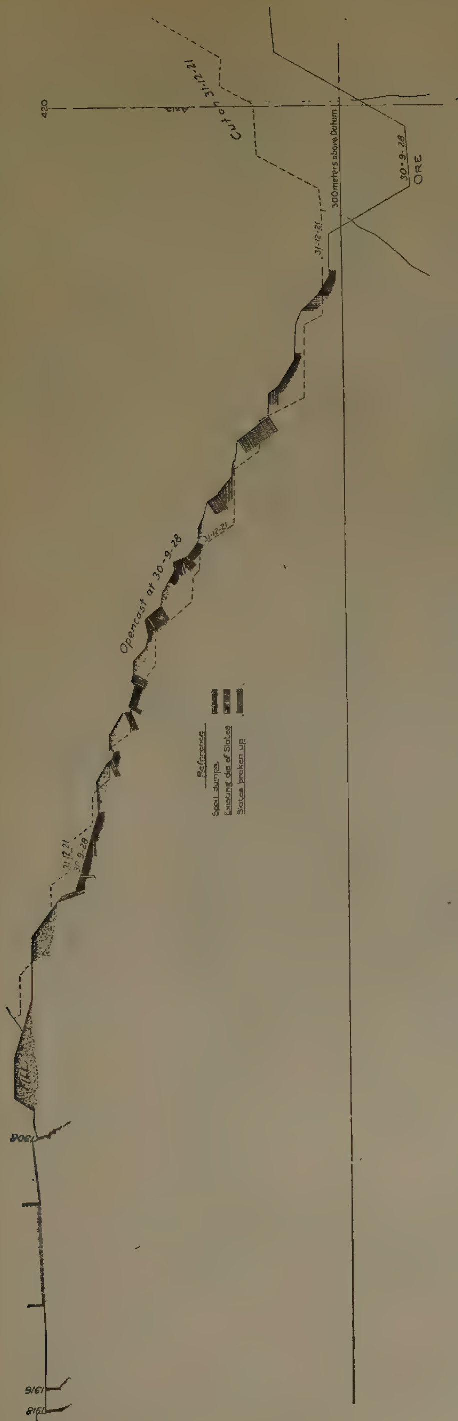

FIG. 14.—ENLARGEMENT OF FIG. 13 TO SHOW FORM OF SURFACE DISPLACEMENT.

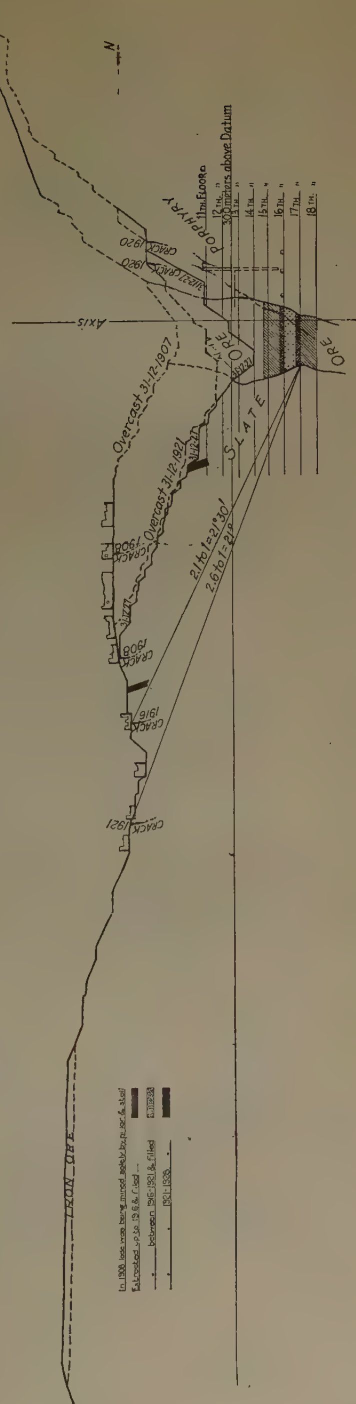

FIG. 15.—SOUTH LODE. CROSS-SECTION ON LINE 360 OF FIG. 16.

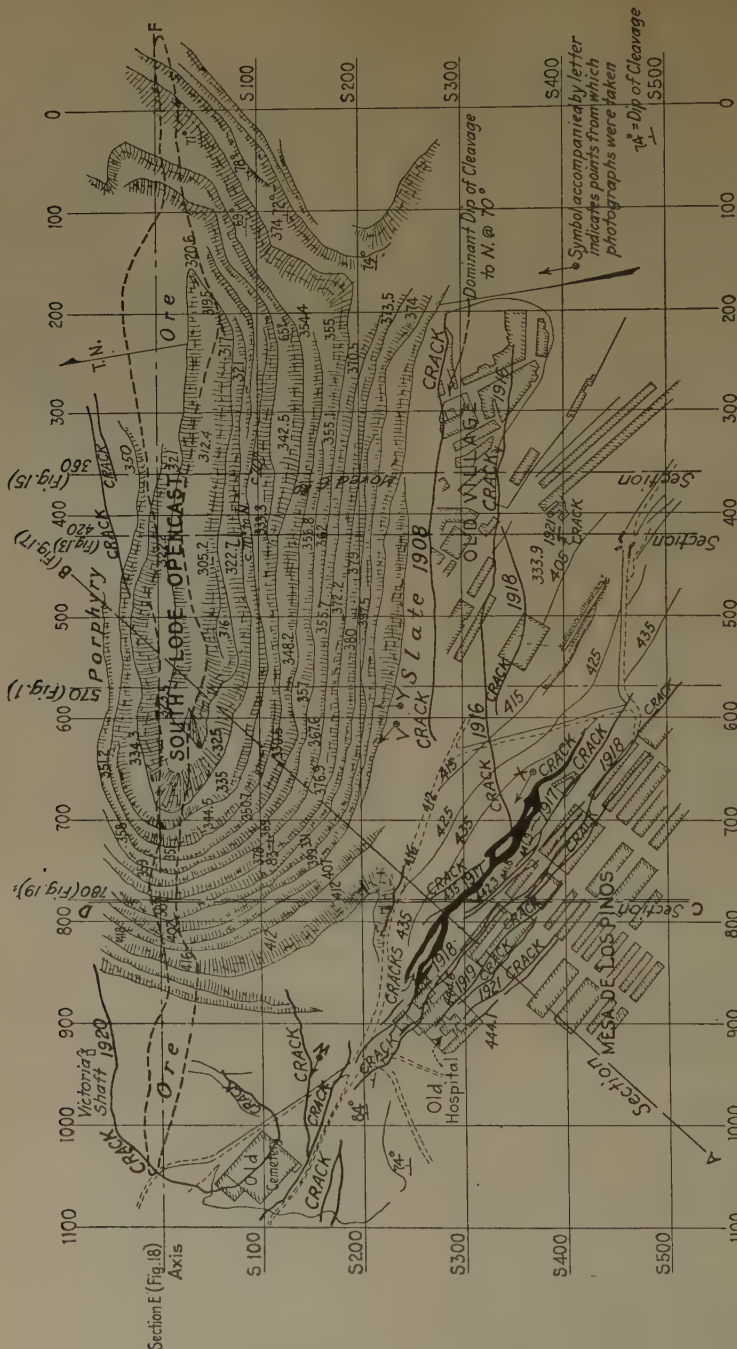

FIG. 16.—SOUTH LODE OPENCAST. PLAN OF SOUTHERN PORTION.

FIG. 17.—SOUTH LODE. CROSS-SECTION ON LINE AB OF FIG. 16.

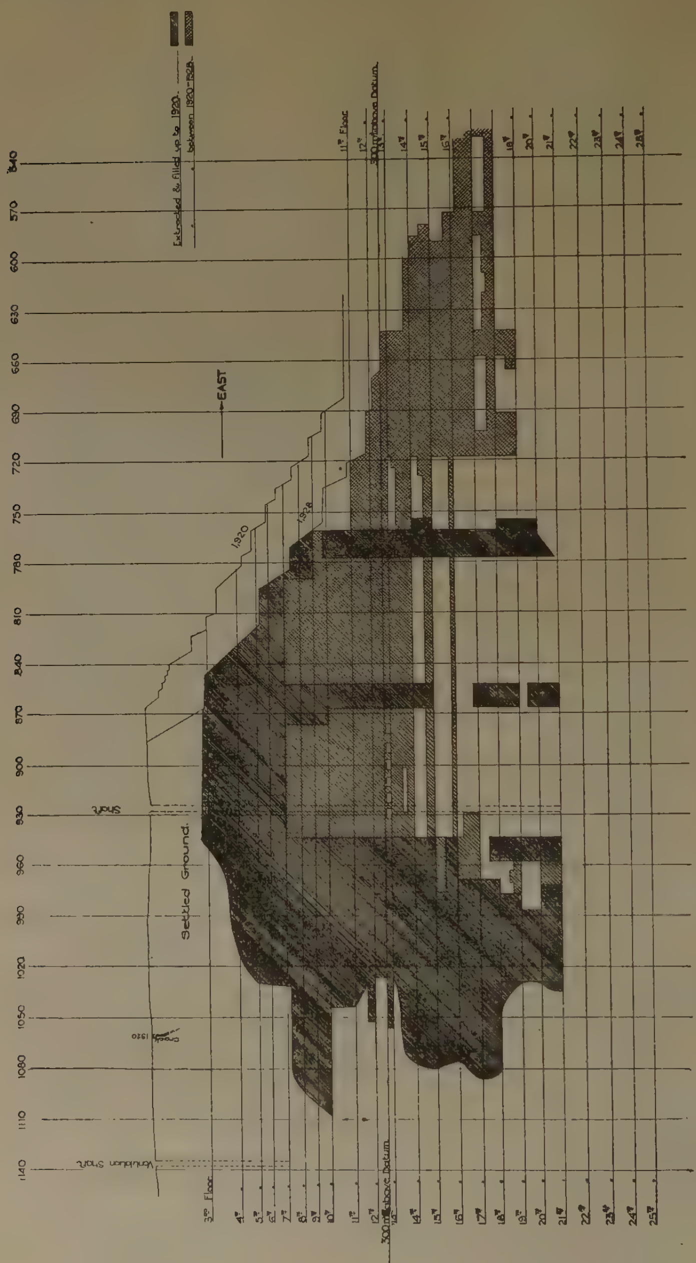

FIG. 18.—SOUTH LODE. LONGITUDINAL SECTION ON LINE EF OF FIG. 16.

stripping benches until the material, requiring a flatter slope when disintegrated, simply conformed to its natural angle of repose. The cracks shown as having opened in 1916 and 1921 have always been a conundrum to the writer, as the ground dropped vertically a meter or so and continued to do so as time went on, leaving a clean break between the moved and unmoved ground and therefore the actual line of fracture must be flatter than that shown by an imaginary line drawn from the excavations to the surface point.

There is no evidence that rain water is trapped in this ground. It enters the cracks during heavy rains, but almost immediately appears

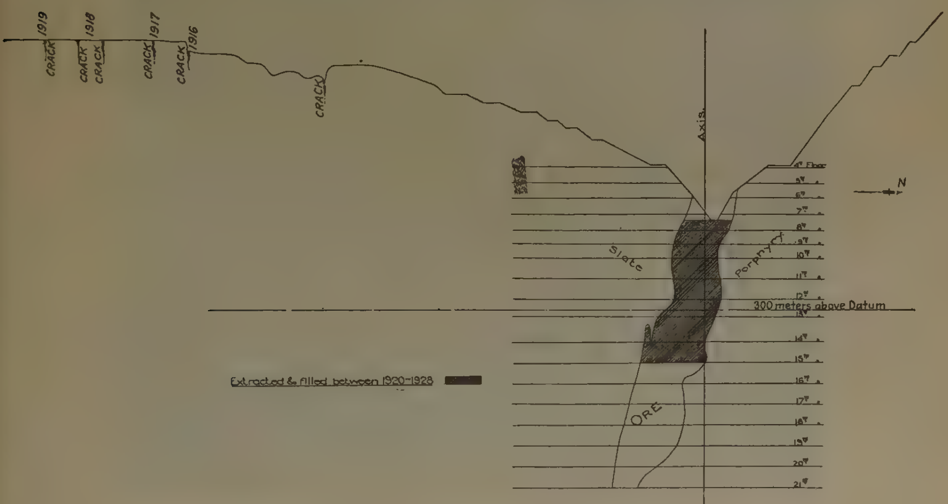

FIG. 19.—SOUTH LODE. CROSS-SECTION ON LINE 780 OF FIG. 16.

and is drained off at the excavation. There are no frosts to assist the movements.

Fig. 16 shows the western end of the pit with the elevations of the various benches and prominent points. The benches are curved, made so for transportation purposes. The sections normal to the axes, although they give a true picture of the lode, do not give a true picture of the stripping slopes.

A heavy ground movement has taken place at the southwest corner, accompanied by a large settlement. The crack marked 1917 opened, broke off in a vertical plane and the portion of the northeasterly side of the break has settled some 3 to 5 m. This crack was followed by others on the dates shown, none of which, however, present such a great settlement as the one that occurred in 1917.

Fig. 20 shows the slate slope of the opencast and Fig. 21 shows the large crack which opened in 1917.

FIG. 20.—SLATE SLOPE OF OPENCAST.

FIG. 21.—LARGE CRACK THAT APPEARED IN 1917

Fig. 17 shows a section taken along the line AB on Fig. 16, normal to the main direction of the crack. This was taken in order to determine, if possible, the locus or cause of these large movements, but, it must be admitted, gives little information. As doubtless the movement is due to excavation in some area between the line of this section and one normal to the axis, a section on this latter plane is included (Fig. 19), taken along line 780 on Fig. 16. The high flat ground on the left-hand side of the section is a plane, said to be the bottom of a geological lake, capped by a bed of limonite 3 to 5 m. thick, overlying the slate. The cracks can be seen extending vertically through the capping into the slates in which they cut the bedding planes diagonally.

A longitudinal section (Fig. 18) showing the excavations gives a more complete view of the picture.

Conclusion

There is little that can be added to the foregoing. The movements recorded do not appear to follow any law and, indeed, are very different in many cases from what might be expected in theory. As careful observations of actual occurrences, however, they are here placed before the profession in the hope that they will add to the sum total of knowledge of the subject and enable some one to correlate them with observations taken elsewhere. Ultimately, perhaps, they may form part of a large mass of similar information from which may arise some set of laws whereby operators in the future may be able to predict ground movements with a reasonable assurance of accuracy.

The Leaching Process at Chuquicamata, Chile

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(New York Meeting, February, 1930)

The ore that is being treated by the present plant lies between the leached zone, or capping, and the mixed sulfide and oxide zone. The principal copper minerals are chalcanthite ($CuSO_4.5H_2O$), brochantite ($CuSO_4.3Cu\ (OH)_2$) and atacamite, ($CuCl_2.3Cu\ (OH)_2$). There are, in addition, some other minerals such as cuprite and kröhnkite ($CuSO_4.Na_2SO_4.2H_2O$). These minerals, with many others, occur in a greatly crushed granodiorite rock.

Most of the copper-bearing minerals occur in the cracks and veinlets, but there are disseminated values and it is probably the relative proportion of these that accounts for some otherwise unexplainable variations in extraction. The entire mineralization of the ore is complex and varies with depth. This variation, the problems presented thereby, and the effect of various constituents of the ore will be taken up in detail.

An analysis of an unweighted composite of the ore treated during 1927 follows:

	Per Cent.		Per Cent.		Per Cent.
Cu	1.58	S	2.10	Mo	0.01
SiO_2	66.10	HNO_3	0.03	As	0.005
Fe	1.41	Cl	0.05	Sb	0.005
CaO	0.20	Na	0.80	Ba	0.01
Al_2O_3	17.70	K	4.80	H_2O	0.70
MgO	0.68	Mn	0.07	O in sulfates	3.94
				Total	100.19

The most important constituents of the ore, as indicated by experience to date, are: (1) The total copper content; (2) the acid-insoluble copper; (3) the acid-making copper mineral, mainly the chalcanthite; (4) the chlorine; (5) the nitrates; (6) the soluble iron; (7) the soluble molybdenum.

The total copper content needs no discussion. In Table 1, the values are weighted averages calculated from individual charge analyses.

The acid-insoluble copper directly affects the extraction, as the present leach recovers practically no acid-insoluble copper. In Table 1 the values are unweighted averages of the analyses of monthly composite samples.

The acid-making copper mineral affects the amount of copper that can be dissolved from the ore without neutralizing acid in the solution. The

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total available acid gained from the ore is determined by the pick-up of $CuSO_4$ by the solutions. In calculating total available acid in solution, twice the copper content plus once the free acid [calculated to H_2SO_4] is taken. In Table 1 the proportions of acid-making copper mineral in the ore are indicated as kilograms of acid gained per metric ton of ore treated. The values tabulated are weighted averages from the monthly total available acid balances. Since the grade of the ore seems to be a direct factor in the acid gained therefrom, two sets of values are shown, the second being adjusted from the first to indicate what the first would probably have been had a constant grade of ore been treated.

The chlorine is important because the chlorine must be removed from the solution before it enters the electrolytic tank house. In Table 1 the values shown are unweighted averages of analyses of monthly composites of the ore.

The nitrates are important because nitric acid affects the materials that can be used in the construction of the solution-handling equipment, the composition of the insoluble anode used in the tank house, oxidizes the iron in the solution to the ferric state, and will itself dissolve cathode copper.

The soluble iron is important because there is a tendency for iron in solution to oxidize to the ferric state under certain conditions and in such state to attack the cathode copper. In Table 1 the values shown are in terms of kilograms of iron gained per metric ton of ore treated and, as in the case of the acid gain, are calculated from solution balances.

The soluble molybdenum is important because molybdenum in solution appears to aggravate the oxidizing effect of the nitric acid on the iron, and to aggravate the effect of the nitric acid itself.

The generally accepted theory of what occurs is that molybdenum in the reduced blue colloidal condition reduces the nitric acid. This would be beneficial if the nitric oxide so produced were evolved from the solution, but most of the nitric oxide produced remains dissolved in the solutions or combines with $FeSO_4$ to form loosely linked compounds such as $2NO : 3\,FeSO_4$. The nitric oxide is reoxidized to form nitric acid in the electrolytic cells and, under such conditions, the oxidation of the iron by the nitric acid, of which there is always a tendency, is greatly speeded up. The ferric iron attacks the cathodes, is reduced to the ferrous state, and is again available for reaction with the nitric acid. The nitric acid is being formed constantly from the nitric oxide through oxidation in the electrolytic cells. Thus it appears that when there is some additional reducing agent present (of the nature of reduced molybdenum), the alternate oxidation and reduction of the iron and nitric acid, during electrolysis, is greatly augmented. It seems to be a series of reactions which, when started, catalyze themselves. Within the past two years unusual activity of the iron and the nitric acid have been traced to rela-

tively high molybdenum in the solution and the relationship has been carefully studied out.

During the year 1929 stabilizing the nitric acid in the solution by treatment with SO_2 has been adopted on a small scale. Apparently the treatment has been of some benefit.

In Table 1 the values of chlorine, nitric acid and molybdenum are all unweighted averages of analyses of monthly composites of the ore treated during the respective periods. The analyses for molybdenum have been made systematically only during the past two years.

TABLE 1.—*Constituents of Ore from Start of Operations to Date*

Year	Ore Treated, Metric Tons	Copper in Ore Treated, Per Cent.	Acid-insoluble Copper in Ore Treated, Per Cent.	Acid Gained per Metric Ton of Ore Treated, Kg.		Chlorine in Ore Treated, Per Cent.	Nitric Acid in Ore Treated, Per Cent.	Iron Gained per Metric Ton of Ore Treated, Kg.	Molybdenum in Ore Treated, Per Cent.
				As Found	Adjusted to 1.60 Per Cent. Ore				
1915	533,328	1.709		2.104	1.973	0.1905	0.1440		
1916	1,562,193	1.660		3.861	3.720	0.1052	0.1440		
1917	2,634,505	1.744		6.668	6.117	0.1011	0.0860		
1918	3,400,634	1.644	0.0334	6.741	6.455	0.1191	0.1028		
1919	2,655,708	1.625	0.0242	6.173	6.075	0.1121	0.0767		
1920	3,847,841	1.524	0.0158	6.172	6.474	0.1167	0.0608		
1921	1,507,651	1.701	0.0391	8.109	7.622	0.1151	0.0932		
1922	3,987,954	1.687	0.0233	8.274	7.845	0.0923	0.0738		
1923	6,400,748	1.663	0.0391	8.746	8.407	0.0932	0.0784		
1924	6,531,345	1.641	0.0400	7.813	7.614	0.0706	0.0382	0.218	
1925	7,055,146	1.565	0.0533	8.959	9.158	0.0518	0.0384	0.230	
1926	7,521,095	1.515	0.0533	8.127	8.572	0.0503	0.0438	0.212	0.0109
1927	6,959,764	1.594	0.0526	7.392	7.415	0.0414	0.0252	0.218	0.0105

	KG. PER METRIC TON ORE TREATED
Iron introduced into solution in dechloridizing	0.013
Iron introduced into solution by anodes	0.008
Total available acid destroyed in dechloridizing	0.700
Total available acid destroyed by electrolysis	0.600

Table 2 gives the analysis of yearly composites of the ore. The results have the advantage of being made by one man and method, but the disadvantage of a very small sample representing a great tonnage of ore, and of not being weighted.

Figs. 1 and 2 were plotted from Table 1. These indicate that the acid-making quality of the ore is diminishing. The percentages of nitric acid and chlorine in the ore are also diminishing, a fact which will in all probability, influence certain changes in the process at some future time. The H_2SO_4 insoluble copper in the ore is undoubtedly increasing and will do so until it is necessary to treat the ore for the sulfide copper.

TABLE 2.—*Analysis of Yearly Composites of Ore*

Year	Total Copper, Per Cent.	Soluble Copper, Per Cent.	Insoluble Copper, Per Cent.	Iron, Per Cent.	Nitric Acid, Per Cent.	Chlorine, Per Cent.	Molybdenum, Per Cent.
1915	1.73	1.68	0.05	0.82	0.39	0.24	0.008
1916	1.71	1.66	0.05	1.01	0.12	0.11	0.009
1917	1.75	1.70	0.05	1.17	0.08	0.06	0.008
1918	1.62	1.56	0.06	1.38	0.10	0.10	0.008
1919	1.61	1.55	0.06	1.57	0.07	0.11	0.008
1920	1.54	1.48	0.06	1.59	0.05	0.11	0.008
1921	1.71	1.66	0.05	1.43	0.05	0.10	0.007
1922	1.67	1.61	0.06	1.38	0.05	0.09	0.008
1923	1.67	1.61	0.06	1.40	0.04	0.08	0.007
1924	1.64	1.58	0.06	1.57	0.03	0.06	0.008
1925	1.59	1.53	0.06	1.52	0.03	0.05	0.008
1926	1.51	1.44	0.07	1.46	0.03	0.05	0.010
1927	1.59	1.52	0.07	1.42	0.03	0.04	0.011
Average.......	1.64	1.58	0.06	1.36	0.08	0.09	0.008

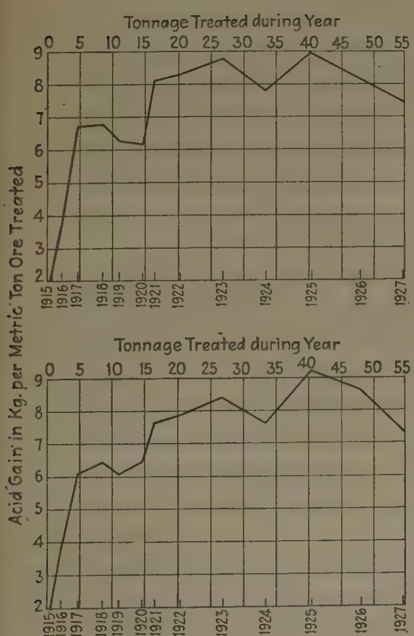


FIG. 1.

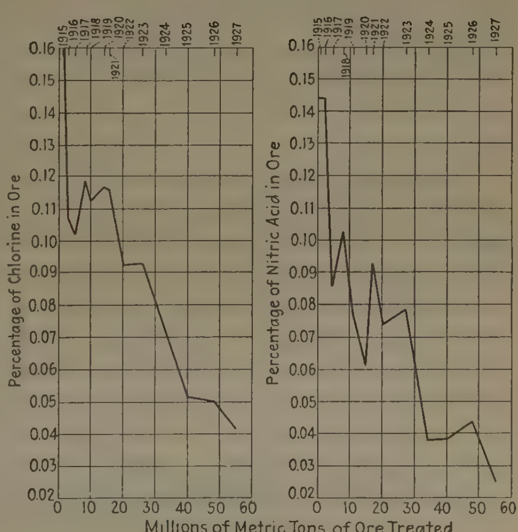


FIG. 2.

FIG. 1.—VARIATION OF TOTAL AVAILABLE ACID GAINED FROM ORE FROM BEGINNING OF OPERATIONS TO JAN. 1, 1928.

Upper curve, values as found; lower curve, values corrected to probable value at 1.60 per cent. copper in ore.

FIG. 2.—VARIATION OF NITRIC ACID AND CHLORINE IN ORE FROM BEGINNING OF OPERATIONS TO JAN. 1, 1928.

Tables 1 and 2 are complete up to Jan. 1, 1928. Up to Jan. 1, 1930, the plant has treated a total of over 80 million tons of ore. During 1928 the acid gain per ton of ore was 7.11 kg. and during the last quarter of 1929 was still as high as 6.79 kg. The chlorine and nitrate contents of the ore have diminished slightly during the past two years. The location of the ores mined with respect to the surface influences the constituents mentioned. Ores from the fringes of the benches are still being mined, a condition that will continue for some time. The proportion of ore exerting a marked influence on the nitric acid, chlorine and water-soluble copper content of the whole is constantly decreasing, but it is not possible to make definite predictions as to when the constituents will change to such an extent as to render advisable or possible any change in the process.

Crushing and Charging the Ore

A flow sheet of the new crushing plant is shown on Fig. 3.

Before the opening of the Chuquicamata plant, it was determined by experiments at Perth Amboy that the maximum efficiency of leaching extraction was obtained when treating ore crushed to ¼-in. mesh. However, with the limitation of equipment in mind (rolls having been installed) it was decided to crush to ½ in. After a considerable period of operation, it was decided that ½ in. particles were too coarse to be penetrated by the solutions in reasonable soaking time, and crushing to 0.371-in. mesh was adopted, the rolls being replaced by disk crushers. This is the present practice. In crushing to this mesh under the present conditions about 10 per cent. oversize remains in the product after screening.

Occasionally there is some trouble caused by the fines produced, which form slimes. These slimes are not only obstinate in treatment in a soaking leach, but prevent even percolation of the solutions, and often cause solution pockets to be retained in the ore after draining. Much consideration has been given to separating the fines and subjecting them to a separate treatment. Screening and handling costs seem prohibitive, considering the relatively little trouble caused by the fines so far.

The present dust-collecting plant was put in operation in January, 1928, and since that time the dust collected from the entire crusher has been charged with the ore going to vats 7 to 13. This amounts to approximately 300 metric tons per day and carries about 4.0 per cent. copper. The results from vats 7 to 13 have been slightly inferior, and those from vats 1 to 6 (receiving no dust) have been slightly better than previously, when the dust was evenly distributed among all the vats. The average remains the same within determinable limits.

The dust is moistened, and so mixed with the ore on the conveyors that it agglomerates well with the coarser ore particles. The ore is also wetted during the crushing process. The moistuse content of the ore as

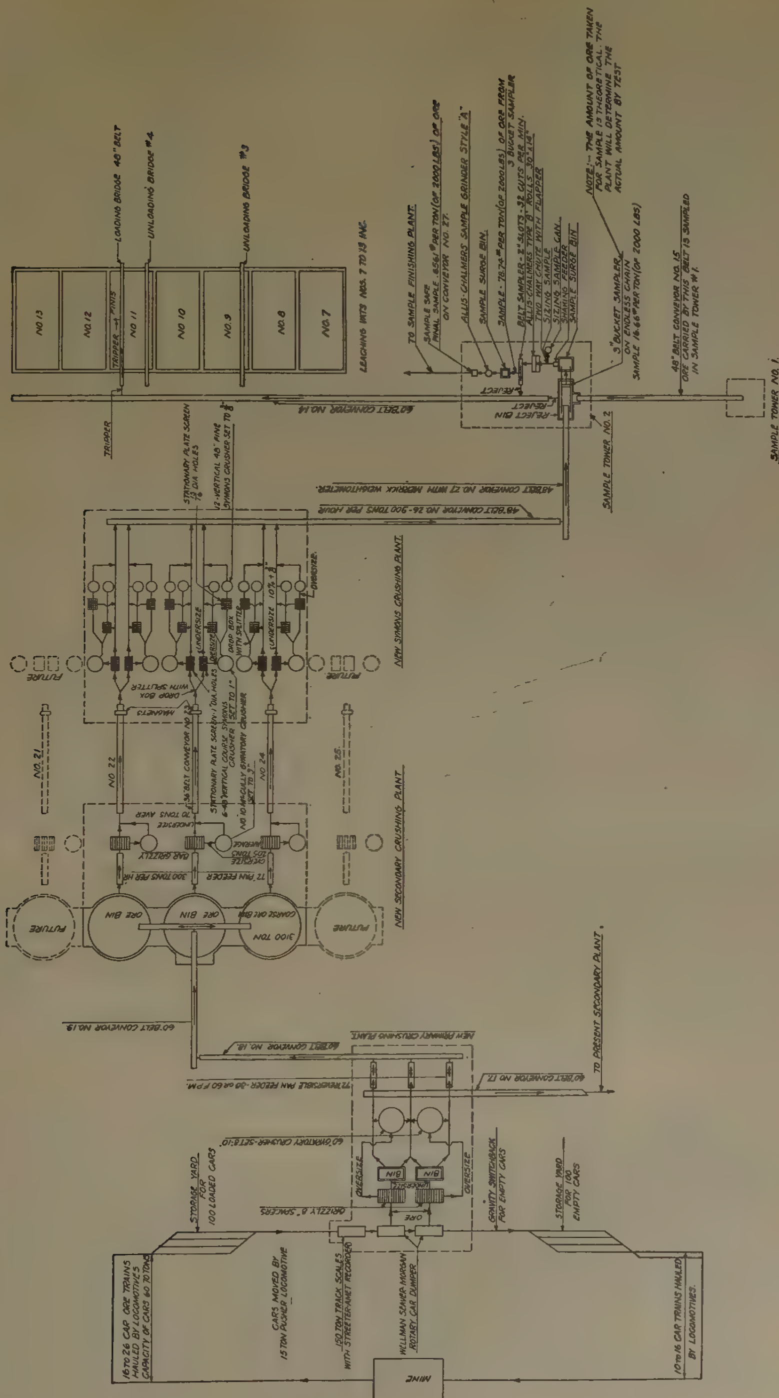


FIG. 3.—FLOW SHEET OF CRUSHING PLANT.

charged into the vats averages 1.85 per cent. No attempt is made to bed the ore while charging. This was tried on a few charges, but results were not encouraging. The vats are filled to the limit as the bridge is moved across.

The stream of ore from the loading bridges plays against a bank of ore at all times, striking near the top. Thus there is some classification of the material as it rolls down the bank of ore. This was not always the practice, but it was found to improve results and was adopted.

Sometimes the ore is charged into solution and sometimes the solution is introduced into the vat after a part or all the ore has been charged. An attempt is made to complete the charging of ore and solution at about the same time to obtain the maximum soaking time. However, the ore is loaded on two shifts only (3 p.m. to 7 a.m.), while the unloading is accomplished from 7 a.m. to 7 p.m., and an attempt is made to avoid crushing and unloading operations on Sundays. Hence the leaching cycle does not follow the same intervals as the loading of the ore at all times. A detailed study of the results from charges loaded in solution and vice versa led to no conclusions, therefore treatment solution is put into the vats whenever produced, regardless of the amount of ore therein.

The vats are loaded to within about 6 in. of the top and the charge leveled by shovelers as soon after loading as possible. Vats 7 to 13 contain approximately an 18-ft. column of material, and vats 1 to 6 a 16-ft. column. The capacity of the vats is about 90 per cent. of the theoretical maximum, due to undischarged tailings. The vats cannot be completely unloaded without considerable damage to the filter bottoms. During the past year, the average charges to the vats were as shown in Table 3.

Table 3.—*Average Charges to Vats during* 1929

	Ore, Metric Tons	Ore, Short Tons
Vats 1 to 6 (called "old type")	9,630	10,615
Vats 7 to 13 (called "new type")	10,840	11,949
Average charge of ore, considering an equal number of both types	10,235	11,282

Outline of Leaching Process

Flow sheets of the leaching and dechloridizing plants and of the electrolytic tank house are given in Figs. 4 and 5.

There are two stages in the process of leaching: (1) that of dissolving the copper from the ore, and (2) that of washing or displacing from the leached ore, or tailings, the dissolved values, or water-soluble copper, that remain in the adhering moisture.

At present, the first stage consists of a countercurrent treatment by two solutions. The first solution on the ore is introduced from the bottom of the vat and percolates upwards through the ore until the latter

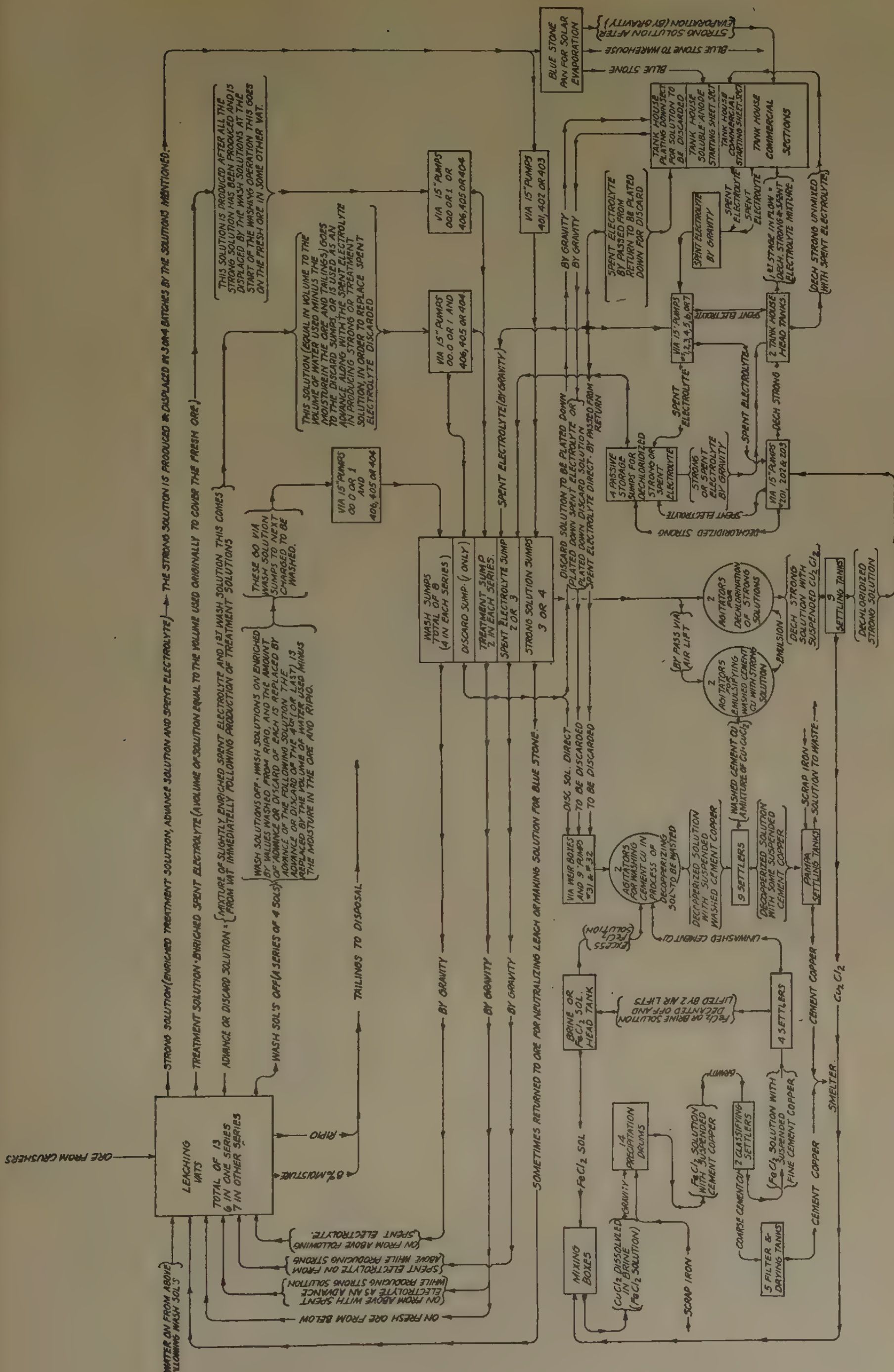


FIG. 4.—FLOW SHEET OF LEACHING AND DECHLORIDIZING PLANTS.

is saturated and covered. This is usually what will be defined as "treatment" solution, and is normally a solution relatively low in copper and high in free acid. After a period of soaking, this solution, or a portion of the amount originally used in covering the ore, is drawn from the

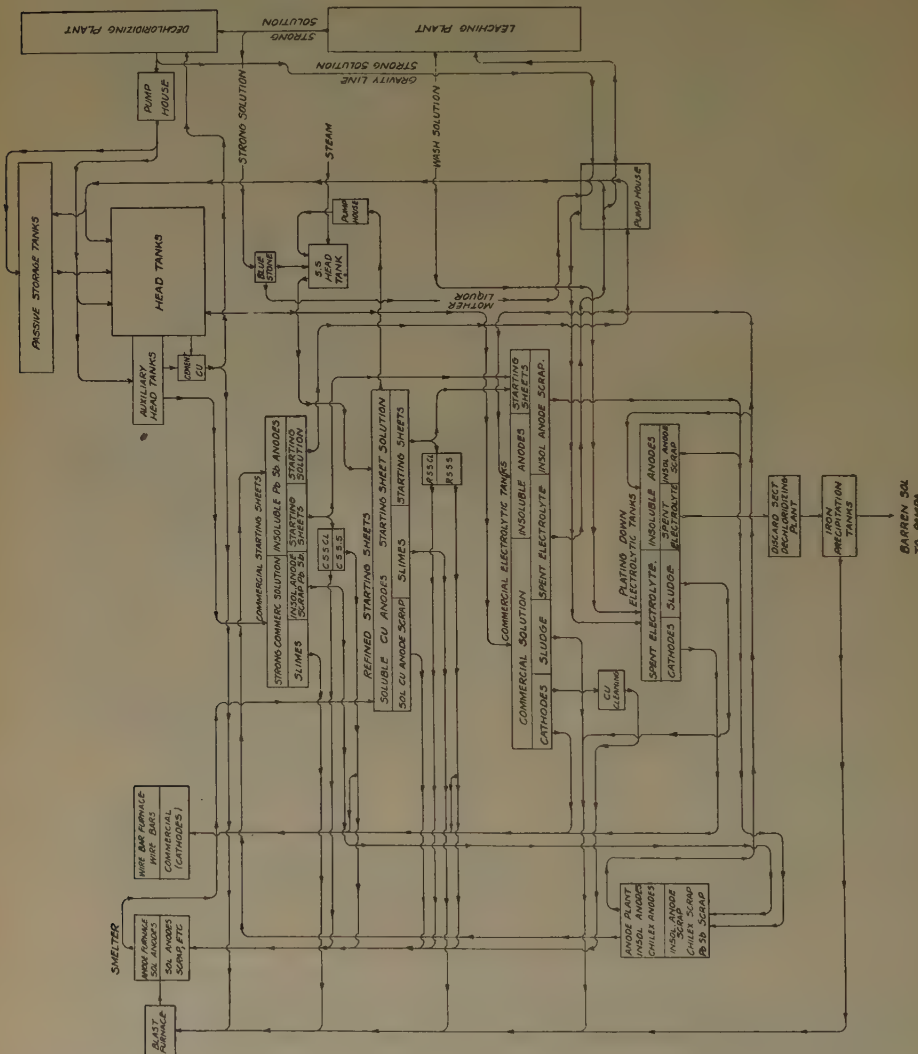


FIG. 5.—FLOW SHEET OF ELECTROLYTIC TANK HOUSE.

bottom of the vat. This solution has been enriched in copper with an accompanying conversion of some of the free acid to $CuSO_4$, and is called strong solution. As the strong solution is withdrawn from the charge of ore, it is replaced, or rather displaced, by spent electrolyte from the tank house. The spent electrolyte is put on the ore at the top of the vat at a rate sufficient to keep the charge covered. It is another solution rela-

tively low in copper and high in free acid. After a certain amount of strong solution has been produced, the process is interrupted. After another soaking period, more strong or enriched solution is produced, being displaced as before by spent electrolyte. Again the process is interrupted, a soaking period intervenes and is followed by a production of another batch of strong, or enriched solution, and displacement by spent electrolyte. It is usual to produce three batches of strong solution from a charge of ore before the final or treatment-producing soak. After the last batch of strong solution has been produced, the charge of ore is allowed to soak in the displacing solutions until it is necessary to wash. The treatment solution is produced during the first part of the washing process, being displaced from the charge by the wash solution coming on. It is used in the treatment of another charge of ore. All solutions in both the leaching process proper and the washing process are withdrawn from the bottom of the vat and all solutions, except that used in first covering the ore, are put on at the top of the vat. Sometimes, to save time, a part of the original cover is put on the top of the vat, but enough is always introduced from below to clear the filter bottom.

The definition of treatment solution that will be followed in this paper is that it is a solution used in the treatment of unleached or partly leached ore and is a solution produced at the start of the washing process, being displaced from, or replaced on the charge, by wash solution. It will also be considered a volume of solution, so produced, equal in amount to the volume of solution originally required to saturate and cover the charge of ore from which it is produced. Frequently more treating solution than is required to cover the charge of ore is produced at the start of the washing process. The amount of this in excess of that required to cover another charge of ore is used with spent electrolyte in producing strong solution, and is called "the advance." The treatment solution, as defined above, can be seen to be spent electrolyte enriched by the ore (the enrichment is slight in the usual method of leaching procedure) and diluted by a certain amount of wash solution. The latter is inadvertently mixed with the treatment solution in displacing the latter from the charge. The advance is diluted treatment solution.

The strong solution will be defined as that solution which, after enrichment by contact with the ore, is sent to the tank house for electrodeposition of part of the copper content; but strong solution must be distinguished from solution sent to the tank house for plating down previous to discard. The latter might be considered to come under the above definition. The content of strong solution depends upon the constituents of the ore, the soaking periods, the content of the solutions from which it was produced, and the order of the batch in production. Batches of strong from one charge are mixed with batches from other charges before sending the strong solution via the dechloridizing plant to the tank house.

In this way, an even grade of solution is usually sent to the tank house. A first strong is normally mixed with a third strong and a portion of either is mixed with a second strong, as required. These strongs are named from the order in which the batches are produced, averaging about 35 grams per liter in copper content. The free acid content will be in relation thereto. During the interruptions between batches of strong solutions produced from any charge, it is the practice to produce batches of strong solution from other charges. A certain order or cycle of procedure is followed in this, to facilitate the mixing of the batches, and to equalize the general rate of strong production and the interruptions for soaking.

Spent electrolyte may be defined as strong solution which, after partial electrodeposition of its copper content, is returned to the leaching plant for re-enrichment. The copper content of the spent electrolyte is normally 15 g.p.l. With consideration of both the leaching plant and the tank house, this was determined as the most economical figure. With the deposition of copper there is a liberation of SO_4 to form H_2SO_4. The H_2SO_4 so formed is reconverted to $CuSO_4$ by the basic copper constituents of the ore. The total available acid in the strong solution and the spent electrolyte is normally about the same. There is, however, a gain of total available acid from the ore. This was mentioned under the discussion of the constituents of the ore.

Sometimes strong solution, or solution produced in the leaching process previous to the production of treatment solution, is returned to ore for further enrichment. To distinguish such solution from strong solution and the treatment solution (as previously defined), solution so produced and returned will be defined as pretreatment solution. Modifications of the present process to include the use of pretreatment solution will be discussed later.

Strong solution, spent electrolyte, treatment solution, pretreatment solution (if any be employed), and all solution passing through the tank house in some intermediate state between strong solution and spent electrolyte, are potentially the same. All these solutions will be considered as forming the primary system.

Post-treatment solution will be defined as solution produced after the production of treatment solution which is not used in washing the leached ore. The advance is post-treatment solution, and the discard, which will be discussed later, usually is. Post-treatment solution becomes part of the primary system if it is advanced.

The time the original covering solution remains on the ore depends more on the ratio of supply of copper from the crusher and the demand from the tank house than on any other consideration. For the best extraction it should be short. Uusually within 6 hr. after the ore has been covered, the copper content of the solution will have risen to about 80 per cent. of the maximum obtainable under the conditions. This maxi-

mum depends on the grade and nature of the ore, and the original copper and free-acid content of the solution. Ores with high acid-making qualities (those with relatively high chalcanthite content) will produce the most rapid initial rise in the copper content of the solution. In any event, the rate of extraction by the treating solution seems to be greatly retarded after this rapid initial rise, and it requires an addition of solution low in copper and high in free acid to speed up the extraction. This speeding up effect is not great, but makes an appreciable difference, especially in a short total soak. However, to replace the treating solution too soon or too frequently will have the effect of diluting the strong solution to an undesirable extent, or of leaving too much copper to be removed by the treatment solution. The procedure, in any case, must be balanced to best fit the conditions of operations and the behavior of the strong solution as it is produced.

The first soak is considered the time of contact of the ore with treating solution previous to removal of any enriched or strong solution. In calculations involving the effect of soaking time, it is figured as one-half the time of originally covering the ore plus the time from the finish of covering to the start of producing strong solution. As previously stated, the ratio of supply and demand usually controls this time. The general procedure is to draw off immediately from a charge the available strong solution, whenever there is any available space in some storage sump for strong solution.

The second soak is considered the time from the start of producing the first strong or enriched solution to the time of finishing the production of the last strong or pretreatment solution. This is a series of periods of replacement of solutions with intervening interruptions. It is in reality several soaks, but is considered as one for purposes of calculations that will be taken up in detail later.

The third soak is considered the time from the finish of the production of the last strong or pretreatment solution to the start of production of treatment solution, or of the start of the washing process, which is the same thing. Two hours are arbitrarily added to this time, since this is one-half the average time required to remove the treatment solution from contact with the ore.

The total effective soaking time, usually called the total soak, is the sum of the three soaks defined above. The length of the total soak depends mainly upon the rate of production. The length of the second soak also depends mainly upon the rate of production. Under most conditions, the second soak is a more or less fixed portion of the total soak (at present approximately one-half thereof). The first and third soaks are also dependent to some extent on the rate of production, but independently of this, the first soak is determined by conditions previously outlined, and the length of the third soak depends on the length of the first

soak. A comparatively long first soak naturally causes a comparatively short third soak, and vice versa.

The curves shown on Fig. 6 illustrate the behavior of strong solution during production. These are based on the strongs produced during the past six months. The contents of the first strong is practically constant until 53 per cent. of the volume covering and saturating the ore has been produced. At this point, the copper content of the solution commences to drop, because the displacing solution is coming through the charge mixed with the enriched solution faster than it is dissolving copper from the ore. After the interruption, the second strong starts off with a copper content a little higher than the finish of the first strong, but this content

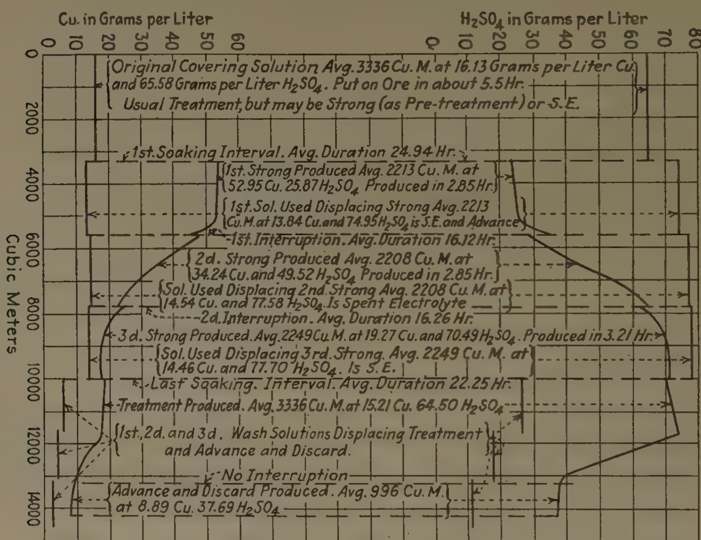


FIG. 6.—BEHAVIOR OF STRONG SOLUTION DURING PRODUCTION.

drops off rapidly, since the values in the solution on the charge appear to remain dispersed after the production of the first strong. There is also a slight pick-up at the start of producing third strong and at the start of the treatment, and both of these solutions appear to have been on the charge in layers with the strongest solution at the bottom of the vat. The drop in the value of the copper in the third strong becomes more gradual as this value approaches that in the spent electrolyte. This strong is interrupted when the copper content reaches 18 g.p.l. which results in the subsequent production of a treatment solution of about the same copper content as spent electrolyte. It can be seen that after the production of second strong, the process is practically a displacing or diluting of the enriched solutions, or a washing process.

At present there are four reasons for producing a low-grade strong solution:

1. The strong solution is produced until a solution that is low in copper content is left in contact with the leached ore at the start of the washing process. This improves the results of washing.

2. Solutions in contact with the ore during the soak, which are low in copper and high in free acid, give the best results in leaching.

3. The lower the content of the strong solution, the less spent electrolyte has to be returned to the head tank for mixing. This mixing is necessary to bring the copper content of the solution entering the electrolytic cells down to a certain value. The more spent electrolyte is returned for mixing, the higher are the ferric iron content and the temperature of the electrolyte, and the lower the power efficiency. The ferric iron in the strong solution has all been reduced in the dechloridizing process and the solution has been somewhat cooled by contact with the ore. This may appear to be somewhat of a burden on the dechloridizing plant, but only under certain conditions. The cement copper gained in the dechloridizing plant must be converted to a wire-bar furnace product, and it is about as economical to put it into the strong solution in reducing ferric iron, and thence into the cathodes, as to convert it into soluble anodes and thence into starting sheets. However, if the reduction of the ferric iron reaches a point where the cement copper required is in excess of the gain over other requirements, it is as economical to reduce the ferric iron with cathodes as with cement copper.

4. It has been found that with relatively high free acid in the strong solution, the molybdenum is not reduced to the colloidal blue state in the dechloridizing process, and that the troubles produced thereby are greatly alleviated. Hence, especially when the molybdenum content of the solutions increases, an effort is made to prolong the production of strong solution from each batch of ore until the average free-acid content of the total strong solution produced is 45 g.p.l. and in some cases as high as 50 g.p.l. It is also important to so mix the strong solutions before dechloridizing that the acid content is always above the minimum requirement.

To produce each batch of strong solution without any premature dilution, the ideal point of cut-off would be after producing 53 per cent. of the covering volume. This necessitates the production of four batches of strong solution before reaching a treatment of the desired grade and obtaining a dilution of the strong to the desired extent. Producing batches of strong equal in volume to approximately two-thirds of the covering volume results in no more than 2 per cent. premature dilution, and the production of a second strong of about the desired content without mixing. Three batches of strong solution of such volume just about meet the requirements, and are easily balanced. Of course, there is a variation in content of the batches due to variations in the grade of ore, but this can usually be taken care of by an adjustment of the volume of the third strong solution alone.

Raising the copper content of the treatment solution by cutting off the last strong solution at a higher value will diminish the volume of the total strong solution produced and increase the copper content thereof. However, when it becomes desirable to accomplish such a result it is possible to do so by the use of a pretreatment solution. A part of the last strong or strongs (volume depending on results desired) can be returned to some subsequent charge of ore, the production as pretreatment solution being continued until the subsequent treatment solution approaches the value of spent electrolyte. Thus the grade of the strong can be raised without increasing the difficulties of washing the leached ore. This use of pretreatment solutions can be extended until a neutral strong solution can be produced, or at least an almost neutral strong. The latter can be neutralized with limestone to throw down the iron, or be treated with SO_2 to reduce the ferric iron. While the strong solution (as previously defined) can be reduced in volume, by the use of pretreatment solution, the total volume of strong and pretreatment solution will be greater than the volume of strong produced without the use of pretreatment solution. This implies, of course, that it is required to produce the same grade of treatment solution in each case. The increase in volume will not be the exact amount of the pretreatment used but will be sufficient to cause serious difficulty in completing neutralization of the strong and the extraction, in the short cycle for which the present plant was designed. This cycle does not obtain at present, but at maximum capacity, the average interval from loading to loading of each vat would be 84 hr. However, it will be possible to adopt a scheme for purification of a part of the strong solution, which could be carried out with a short cycle, and not seriously affect the extraction.

The fact has been mentioned that the division of the soaking time and the free-acid and copper content of the solutions in contact with the ore have an effect on the extraction. From a detailed study of the behavior of the solutions during the leaching process, and complicated calculations made from the results obtained from a considerable number of charges treated, the following formulas were deduced:

A = free acid in first solution on ore, g.p.l.
B = free acid in start of first strong produced, g.p.l.
C = average free acid in solutions displacing strong and pretreatment, g.p.l.
D = average free acid in strong and pretreatment solution produced g.p.l.
E = free acid in last solution used to displace strong or pretreatment solution, g.p.l.
F = free acid in solution last covering ore before washing, g.p.l.
G = copper in first solution on ore, g.p.l.
H = copper in start of first strong produced, g.p.l.

I = average copper in solutions displacing strong and pretreatment solution, g.p.l.

J = average copper in strong and pretreatment solution produced, g.p.l.

K = copper in last solution used to displace strong or pretreatment solution, g.p.l.

L = copper in solution last covering ore before washing, g.p.l.

R = first soak, hours.

S = second soak, hours.

T = third soak, hours.

U = total soak = $R + S + T$

X = average free acid in solution in contact with ore during soak, g.p.l.

Y = average copper in solution in contact with ore during soak, g.p.l.

$$X = \frac{(2B + A)R + (2D + C)S + (2F + E)T}{3U}$$

$$Y = \frac{(2H + G)R + (2J + I)S + (2L + K)T}{3U}$$

These equations are not exact, and the quantities X and Y obtained are factors rather than exact constituents of the solutions. However,

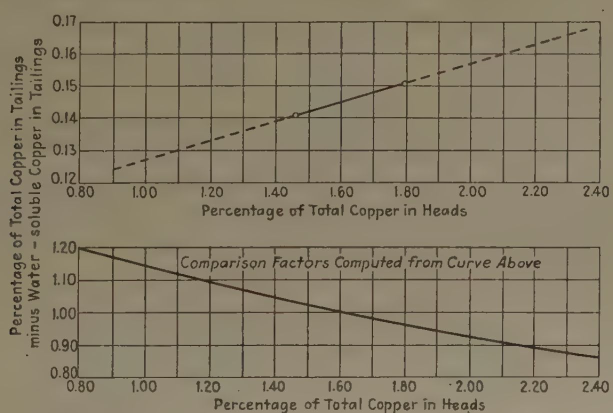


Fig. 7.—Relation of non-water-soluble copper in tailings to total copper in heads.

Based on 90 hr. total net soaking time; 55 g.p.l. H_2SO_4 average free acid in solutions in contact with ore during soak; 0.053 per cent. acid-insoluble copper in heads; 0.181 per cent. oversize on 0.371 in. in heads.

Lower curve gives comparison factors computed from upper curve. Multiply non-water-soluble copper in tailings, obtained under any conditions, by factor corresponding to percentage of copper in heads, to obtain probable value of non-water-soluble copper in tailings had the heads contained 1.60 per cent. copper and other conditions remained the same.

these factors are relative (within satisfactory limits) in their application to determine certain results expected from variations in the leaching process. X has a bearing on the results of leaching or on the amount of

non-water-soluble copper in the tailings. *Y* has a bearing on the results of the washing, or on the amount of water-soluble copper in the tailings.

The effects of the grade of the ore, the length of the total soak and average free-acid content of the solutions in contact with the ore during the soak on the amount of non-water-soluble copper in the tailings are shown on Figs. 7, 8 and 9. The curves were computed from a detailed analysis of the results obtained from the treatment of over 3,500,000 metric tons of ore. Even so, they are merely indicative of what may

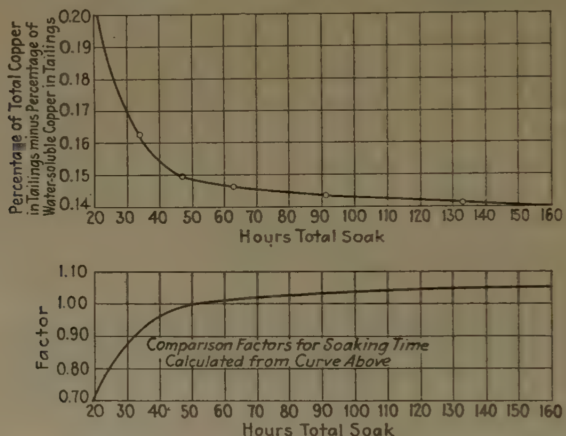


FIG. 8.—RELATION OF NON-WATER-SOLUBLE COPPER IN TAILINGS TO TOTAL SOAKING TIME.

Based on 1.60 per cent. copper in heads; 55 g.p.l. H_2SO_4 average free acid in solutions in contact with ore during soak; 0.053 per cent. H_2SO_4 insoluble copper in heads; 9.181 per cent. oversize on 0.0371 in. in ore.

Lower curve gives comparison factors for soaking time calculated from upper curve. Multiply non-water-soluble copper in tailings, obtained under any conditions, by factor corresponding to hours total soak, to obtain non-water-soluble copper in tailings that would probably have been obtained with a 50-hr. soak, other conditions not being changed.

be expected over a long period of production, since the results from individual charges vary widely under identical conditions.

Outline of Washing Process and Discarding

The treatment solution usually is the first solution brought in contact with the ore, although the first solution may be pretreatment solution or more often spent electrolyte. In any event, treatment solution is the solution in contact with the ore previous to washing and the copper content of this solution is one of the biggest factors in the efficiency of the washing process.

The washing process consists of a downward displacement of the solution covering and saturating the leached ore or tailings by a series of wash solutions and water. If this covering solution—treatment solution—were withdrawn without displacement, the volume removed would not be equal

to the volume originally covering the ore, since the tailings retain about 8.25 per cent. of their weight in moisture. Approximately one-quarter of the original volume of the solution covering the ore is retained by the tailings and the problem of washing is to displace and dilute the solution so retained. In practice, the washing of Chuquicamata ores has not proved to be a true piston displacement of solutions, or even a series of complete dilutions. The solution not fully absorbed by the tailings is displaced with a certain amount of mixing, but the solution absorbed by the tailings does not become thoroughly mixed with the displacing solutions at the rate at which they pass through the vat. If sufficient solution and water could be used, and a slow enough wash given, the values in the final tailings moisture could be reduced almost to nothing. As it is, these values are reduced to economic limits.

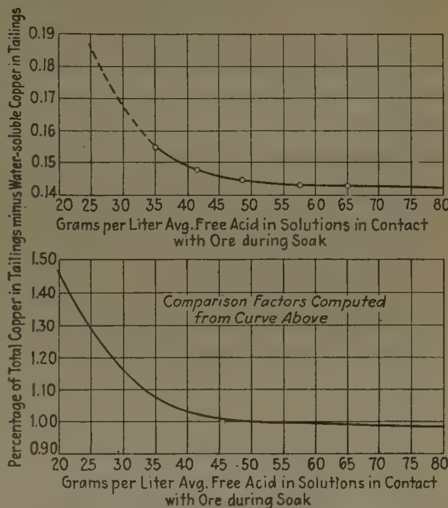


FIG. 9.—RELATION OF NON-WATER-SOLUBLE COPPER IN TAILINGS TO AVERAGE FREE H_2SO_4 IN SOLUTION IN CONTACT WITH ORE DURING SOAK.

Based on 90 hr. total soak; 1.60 per cent. copper in heads; 0.053 per cent. acid-insoluble copper in heads; 9.181 per cent. oversize on 0.371 in. in heads.

Lower curve gives comparison factors computed from upper curve. Divide non-water-soluble copper in tailings, obtained under any conditions, by factor corresponding to average free acid in soak, to obtain probable value of non-water-soluble copper in tailings, had the average free acid in soak been 50 g.p.l. and other conditions remained the same.

The wash solutions are in reality a mixture of treatment solution and water, the percentage of treatment solution decreasing and the percentage of water increasing throughout the series. Water alone forms the last wash of the series. To keep the wash solutions from building up in copper content, as they are used over and over again from charge to charge, a certain portion of the first wash is advanced, or discarded, and replaced by an equal amount of second wash. This amount of second wash is replaced by third wash and of third wash by fourth. The amount of fourth wash so advanced is replaced by water. To bring about a volumetric balance, enough water is used to replace the advance of fourth wash and satisfy the tailings moisture.

The amount of first wash advanced either becomes the advance, as previously defined, or is pumped directly to the discard storage sump. Often part of it is used as advance and the remainder goes to the discard sump. In any case, it is classified, measured and sampled as a separate

unit. This solution, no matter what its subsequent destination, comes under the classification of post-treatment solution, as previously defined. The volume of this solution depends on the volume of water used, being increased directly per unit of increase of water. The increase in total content of this solution does not increase directly with the increase of water, since part of this water dilutes the post-treatment solution. This is why the total amount of copper, total available acid and other constituents in the solution to be discarded are not greatly influenced by the volume of water used when making a post-treatment discard. On the other hand, when spent electrolyte is discarded, the post-treatment solution is used as the advance to make up the volume of solution discarded from the primary solution system. In this case, both the amount and total content of the solution discarded increase directly as the amount of water used, until a certain balance is reached.

No matter where the discard of solution is made, or the volume of solution discarded, the total amount of any constituent discarded must eventually equal the gain of this constituent from the ore. In making the preceding statement, the copper and free acid are considered as parts of the same constituent (the total available acid), since copper removed electrolytically releases free acid in a certain fixed proportion. Even so, this statement is only approximately true, since a certain amount of total available acid is destroyed in the tank house and the dechloridizing process. However, the amount discarded or otherwise removed must equal the gain from the ore, and the amount of total available acid otherwise removed does not make any difference in the amount discarded in relation to the point of discard or the amount of water used. Some iron is introduced in the tank house and in dechloridizing the strong solution, but the amount is small in proportion to the amount introduced by the ore, and as in the case of the total available acid, does not enter the point under consideration. The chlorine, of course, is mainly removed otherwise. The point is this. The constituents of the primary solution system will come to a balance when the amounts removed therefrom equal the amounts introduced. Since all constituents of the primary system, with the exception of the chlorine, are distributed throughout this system in approximately the same proportion as the total available acid, the discussion will be limited for the most part to this constituent.

When discarding spent electrolyte, the total available acid in the primary system must be diluted to the same content per unit volume as the solution discarded. This effect is often obscured because the total volume of the primary system is large and it requires a long time for the constituents thereof to come to a balance under any one set of conditions. However, the eventual effect of increasing the amount of water used, when discarding spent electrolyte, is to increase the dilution of the

entire primary system and to the same extent the discard, but not to increase the total amount of total available acid discarded.

When discarding post-treatment solution, the solution discarded is diluted directly and the concentration of total available acid in the primary system is not materially affected by the amount of water used, or, therefore, by the amount of this solution discarded, although in either form of discard the total amount of total available acid discarded will be the same, after the solution system has come to balance.

For the preceding reasons, when it is desirable to increase the total available acid in the primary system, post-treatment solution is discarded, and to diminish the total available acid in this system, spent electrolyte is discarded. In the latter case, the amount of water must be controlled carefully; in the former, it is not so important. To diminish the concentration of iron, nitric acid and molybdenum in the primary system, a spent electrolyte discard is used. To accomplish this last result, a certain amount of total available acid concentration must be sacrificed. Since the introduction of the Chilex anode and the 1922 modification of the dechloridizing process, it has been practicable to control the constituents of the primary solution system by the solution discard alone and without the addition of commercial H_2SO_4. This may not always be possible and there are times at present when this is not entirely satisfactory. However, no form of removing the iron, nitric acid or molybdenum, other than by solution discard, has been incorporated into the regular process.

In displacing the treatment solution from the charge, there is a certain inevitable advance of first wash solution into the primary system, due to the mixing during displacement and the retention of solution as moisture by the trailings. Thus the treatment solution produced is not exactly the same as that solution in contact with the tailings just before washing. It is the latter solution that really influences the values carried into the post-treatment, the washes and by the final moisture in the tailings. For each charge treated, samples are taken to determine the contents of this solution, which is called the "last cover before washing." This is the primary solution that must be washed from the tailings. The amount of this solution retarded and the amount of first wash inadvertently advanced are equal, the one simply being displaced by the other. This amount is independent of the amount of water or wash solution used, and is somewhat less than the amount of moisture retained by the tailings, although it is a function of the latter. It averages about 650 cu. m. per charge, or 635 liters per ton of ore treated.

The primary solution retarded is distributed through the post-treatment, the washes and the final moisture in the tailings, bringing the values in the wash solutions to a certain balance depending on the

amount of water used and the point of discard. The amount of dilution by this inevitable advance to the primary system depends on the portion of water in the first wash. The first wash is somewhat diluted by an increase in the water used, but the dilution is not nearly in proportion to the water; hence neither is the dilution in the primary system when no deliberate advance is employed. To discard post-treatment solution produced at any other point than between treatment and first wash (*i. e.*, discarding after the production of first, second, third or fourth wash) will not aid materially in the concentration of values in the primary system. The amount of primary solution retarded will be the same in any case. The washes preceding the point of discard simply become a part of the primary system, increasing in concentration until they carry practically as much of any constituent per unit of volume, back into the discard, washes following the discard, and the final tailings moisture, as would be carried back by the primary solution in contact with the tailings before washing. Continuing a discard of solution following any wash has produced the same results as though that wash and those preceding had been eliminated. The real effect of setting back the point of discard in the wash system is to increase the proportion of primary solution in the final tailings moisture and decrease the proportion in the direct solution discard. This is not economical, since the former cannot be decopperized before discarding and the latter can. In connection with this point, it must be remembered that the total amount of primary solution discarded is the amount in the deliberate solution discard and the amount in the final tailings moisture, no matter whether post-treatment solution or spent electrolyte is discarded.

The concentration at which any constituent will come to a balance in the primary system can be calculated, as follows: To the amount of any constituent gained by solution from the ore, add any amounts of this constituent otherwise introduced and subtract amounts removed or destroyed by means other than solution discard. The resulting amount should be calculated in terms of grams gained per metric ton of ore treated. Divide this result by the liters of primary solution discarded per metric ton of ore treated and the quotient will be the concentration, in grams per liter of the constituent in the primary system, that will result from any fixed amount and form of discard. Certain calculations that will appear later are based on calculation, made in this manner.

To prevent confusion, the following point was not previously mentioned. There is a certain amount of inevitable advance of water into the primary solution system with each charge treated, due to the moisture in the original ore and the tailings left in the vat after unloading. Since this amount is almost constant in amount per charge treated, it does not enter into the computation of the amount of treatment that it is necessary to produce or to cover a charge.

In conclusion, the point and amount of discard are determined with regard to the relative concentration of total available acid, iron, nitric acid and molybdenum in the primary system, remembering, of course, that using a post-treatment the amount of discard is not so important, and that the more water is used, the better the result of washing the tailings will be. The discard will have to be adjusted to meet variations in the relative gain of the previously mentioned constituents from the ore. The total available acid gain is usually the dominating factor in this respect but the problem of decopperizing the discard solution must be considered.

In the dechloridizing plant, the copper removed from the discard solution is converted into cement copper. It is not economical to produce more than a certain amount of this, therefore this consideration limits the copper content of the solution to be discarded. It has been previously shown that, no matter where the point of discard or what the volume thereof may be, the total amount of total available acid destroyed and discarded must eventually equal the gain thereof from the ore. The concentration of total available acid in the primary system may be influenced, but not the amount discarded. The portion of copper in the total available acid will vary considerably; if the discard is spent electrolyte, the copper content will be that of the return from the tank house, normally 15 g.p.l., but if the discard is post-treatment solution, the copper content will normally be much less. It will depend on the copper content in the last cover before washing, and to a lesser extent on the conditions during the soaking periods and the relative lengths of the soaks. This last factor, as will be explained, can be expressed in terms of the average copper content of the solutions in contact with the ore during the total soak. The amount of water used influences the total copper in the solution discarded only to the extent that it diminishes or increases the amount of copper retained by the final moisture in the tailings. This is relatively negligible under most conditions. The total copper content and the relative concentration in grams per liter must not be confused. The concentration is influenced by the amount of water used far more than the total content. In connection with this point, it is necessary to explain why the total available acid is always apparently less in a post-treatment discard than in a discard of spent electrolyte, in the face of the statement that the total amount of total available acid will eventually be the same in either case. The explanation lies in the word eventually. It is not desirable to continue with as large a discard of spent electrolyte as of post-treatment solution, in consideration of the dilution of the primary system; thus it is usually possible to dilute the post-treatment discard more than the spent electrolyte discard. Regardless of dilution, the copper content of a post-treatment discard is about 70 per cent. of the content of a spent electrolyte discard under the present method of producing strong and treatment solutions. The present method, as has been

explained, permits the production of a last cover before washing of copper content very little in excess of the copper content of spent electrolyte. The post-treatment solution, however, is usually diluted to from 8 to 10 g.p.l. in copper content.

It is the total copper content that influences the amount of cement copper produced in the dechloridizing plant. To reduce this, it is the practice to send the solution for discard to the tank house for partial plating down previous to the final decopperization in the dechloridizing plant. By plating down is meant partial electrolytic decopperization. It is not as economical of power to plate down discard solution as to plate down strong solution, so the tank house sacrifices some capacity in plating down solution. It is not economical to plate down solution much lower than 7 g.p.l., although cathodes suitable for treatment in the wire bar furnace can possibly be produced when plating down as far as 5 g.p.l. As a rule, post-treatment solution is not plated down but goes directly from the discard storage sump to the dechloridizing plant. Spent electrolyte for discard is usually plated down and in such condition is returned from the tank house to the dechloridizing plant. However, at full production it may prove necessary to plate down either solution to keep the production of cement copper down to the minimum requirement. In such case, the total copper content and not the concentration of copper in the discard solution must be considered. It must be remembered that plating down in no way affects the total available acid discarded. With the plating down of copper, free acid is liberated.

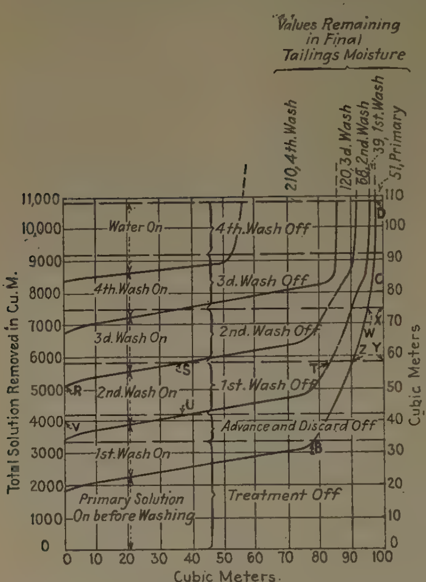


FIG. 10.—DISPOSITION OF SOLUTIONS IN WASHING WITH 1650 CU. M. EACH OF FOUR WASHES AND 1700 CU. M. OF WATER.

Original volume of cover, 3400 cu. m.; moisture in tailings, 875 cu. m.; washing rate, 800 cu. m. per hour.

Solutions off are represented by areas between horizontal lines and vertical lines 0 and 100. Amount of each solution on, in any solution coming off, is represented by the area between the curves indicated, within the area representing the solution off. Each small square represents 100 cu. m.

Examples.—Area R-S-T-U-V = amount of second wash on that comes off in the first wash = 1009 cu. m.

Area W-X-Y-Z = amount of primary solution on before washing, coming off in second wash = 120 cu. m.

Some of the preceding points may be clarified by a study of Fig. 10. This diagram was constructed from the results calculated from over 300

charge records, and in addition data were obtained from 15-min. samples of solutions from charges for which the wash solutions had been equalized by mixing previous to their application.

The curve bounding the treatment or primary solution closely follows a certain equation from points b to c. From the origin of the curve to point b it is almost a straight line. This portion of the curve denotes the mechanical mixing of solutions not saturating the ore. From points c to d, the curve flattens out on account of a change of action during the process of drawing down for the final water wash and of draining.

The amount of primary solution remaining in the vat after removal of any amount of solution can be roughly computed by the following equation:

$$R = \frac{M}{C^v}$$

R = cubic meters of primary solution remaining in the vat.
M = cubic meters of moisture remaining in the tailings.
C = a constant, depending on nature of ore and washing rate.

An average value for C, which holds approximately for washing rates from 400 to 1200 cu. m. per hr., is:

$$C = 1.440 - 0.00004W$$

where

W = rate of washing in cubic meters per hour.
$v = U - D$
U = cubic meters solution removed after start of washing.
$D = T - M$ = cubic meters of drains.
T = cubic meters solution covering and saturating leached ore before washing.

From the origin of the curve to point b, the amount R may be calculated as if the curve were a straight line from $U = D - 700$ to $U = D + 750$. Point b is at $U = D + 750$. This is not exact, and the quantity of R so calculated must be adjusted to equal that calculated at $U = D + 750$ by the equation. The other curves are practically parallel to the one discussed, and follow the same at intervals representing the volumes of the washes. With this in mind, values may be calculated by the equation for each of the washes.

The amounts so calculated of primary solution and each of the washes, as coming off in the last wash, should be increased by 20 per cent. to correct for the change of action during the drawing down for water and draining. The amounts of increase should be deducted from the respective amounts of each solution coming off in the next to last wash.

From the equation or diagram, the amounts of primary solutions and water, composing the treatment off, the advance and discard, and each

of the washes can be calculated, presuming that washing has been conducted under the same conditions until the system of solutions has attained a balance. To use the diagram for other amounts of water than 1700 cu. m., or points of discard other than that indicated, the horizontal lines will have to be shifted accordingly.

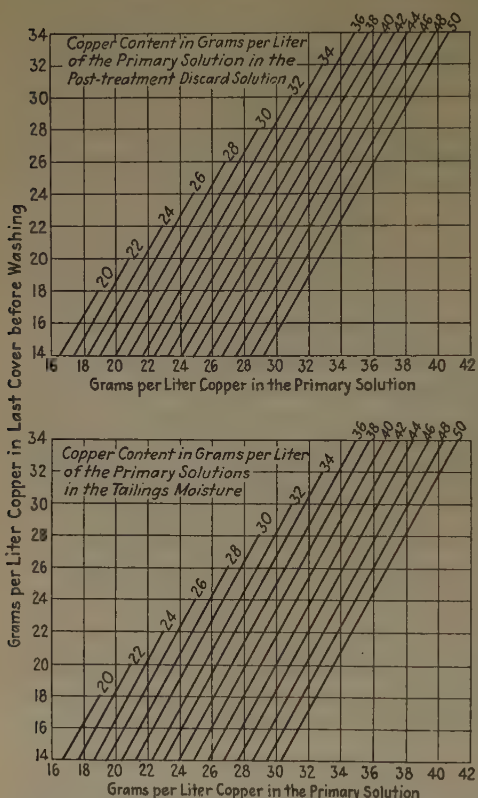


Fig. 11.—Effect of average copper content of solutions on copper content of advance and discard and tailings moisture.

The number at top of each line equals grams per liter copper average of solutions in contact with ore during soak.

It must be remembered that the point c will remain at the same vertical distance from point d under any conditions. This distance is equal to the amount representing volume D.

While the diagram or calculations by the equation are not exactly accurate, the results from operations for the last six months were closely checked thereby.

In constructing the diagram and making calculations, the total available acid in the solutions rather than the copper content was traced. Due to the layer effect, previously described, the copper content of the primary solution forming the last cover before washing is varied throughout the charge. Also, the amount of copper in that portion of solution saturating the tailings as moisture is somewhat higher than the apparent copper content of the solution forming the last cover. This is due to a lack of thorough mixing throughout the charge, especially in the particles of ore or tailings. It must be remembered that solution penetrating the particles during 30 to 80 hr. of soaking is not readily mixed with solution subsequently brought in contact therewith. The effect of this lack of mixing becomes more pronounced as the last portion of the soak is shortened. Invariable copper gains in the water-soluble and wash balance for practically all charges, and the content of the wash solutions, demonstrate this fact. A relation between the copper content of the primary solution coming off in the advance and discard and in the tailings

moisture and the average copper content of the solutions in contact with the ore during the soak has been traced. The effect of the latter, combined with the effect of the copper content of the solution forming the last cover before washing, on the content of copper in the primary

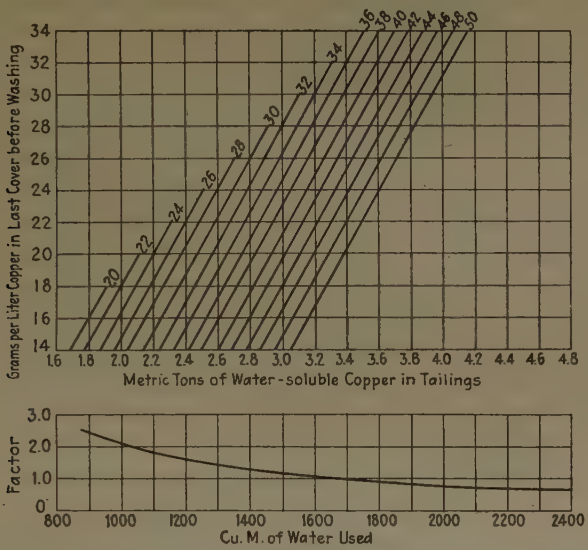


FIG. 12.—VARIATION OF WATER-SOLUBLE COPPER IN TAILINGS.

Based on 10,235 m.t. ore in charge; 875 cu. m., or 8.26 per cent. water in tailings; 1700 cu. m. of water used; washing rate 800 cu. m. per hour. To translate values to percentages, divide by 97.23.

The number at top of each line equals grams per liter copper average of solutions in contact with ore during soak.

Values depend on use of 6600 cu. m. of wash solution, or 645 liters per metric ton of ore.

Lower curve shows factors for approximate corrections to upper figure for volumes of water used other than 1700 cu. m. Multiply results from upper chart by factor corresponding to volume of water used, as found from lower curve.

TABLE FOR CORRECTION OF VALUES

Found from This Chart and Curves on Fig. 13

Washing Rate, Cu. M. per Hr.	Multiply Results by	Washing Rate, Cu. M. per Hr.	Multiply Results by
1,200	1.0533	700	0.987
1,100	1.0400	600	0.973
1,000	1.0267	500	0.960
900	1.0133	400	0.947
800	1.0000		

solution composing a portion of the advance and discard and the tailings moisture, is shown on Fig. 11. The effect of the content of the solutions under discussion on the water-soluble copper content of the tailings is shown in metric tons on Fig. 12.

Fig. 13 shows the amounts of primary solution and water composing the advance and discard and the final tailings moisture; also the amounts

of these in the total discard. From the diagrams on Fig. 11 and the curves on Fig. 13 the copper content of the advance and discard and of the tailings moisture can be calculated.

Fig. 12 shows the effect of some influencing factors on the water-soluble copper of the tailings, but the general character of the ore or

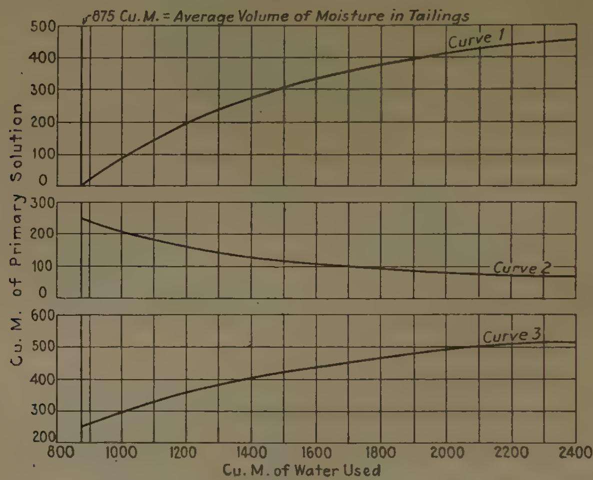


FIG. 13.—VOLUME OF PRIMARY SOLUTIONS.

Curve 1 shows volume of primary solution in discard solution with various amounts of water, when moisture in tailings = 875 cu. m. and discard is between treatment and first wash. This is the advance, when discarding spent electrolyte. To get total volume of solution, subtract 875 cu. m. from cubic meters of water used. To get volume of water in solution, subtract values found from this curve from total volume of solution.

Curve 2 shows volume of primary solution in tailings moisture in same conditions as for curve 1. Also applies when discarding spent electrolyte, and should be added to spent electrolyte discard to obtain total primary discard.

Curve 3 shows total primary discard, when discarding under conditions of curve 1. This is sum of curves 1 and 2, and is also the volume of water advanced to primary system when making no deliberate advance.

To get total volume of water advanced, when making deliberate advance, add values from this curve to value of water in advance, determined from curve 1.

Values shown by these curves are cubic meters per charge treated. To translate to liters per metric ton ore treated, divide by 10.235 the cubic meters found from curves.

Curves are based on use of 6600 cu. m. of wash solution, or 645 liters per metric ton of ore.

tailings often greatly overshadows the factors contemplated. The results indicated on this figure check very closely with average results calculated over 300 charge records.

The benefit of a slower washing rate does not seem to be as great as would be anticipated. The increase in the primary solution displaced from the tailings and entering the washes in large part goes back into the tailings through the increased content of the washes. The volume of wash solution must increase with the time of washing to get the full effect

of increased contact time. Since the volumes of wash solution now used are limited by sump space, to augment these further will entail the establishment of a countercurrent wash system. A few simple pipe-line changes would make this possible even with increased production, and it is possible now with the present installation. However, once such a system is established a very careful regulation of the cycle must be maintained, and the leaching plant cannot serve, to the extent that it does at present, as the buffer in absorbing irregularities of operation. Such a scheme is worth contemplation and a trial under the present conditions of decreased production.

Due to the size and length of the pipe lines, a lineful of strong solution or spent electrolyte, mixing with the wash solutions in the process of washing a charge, will considerably affect the results from that charge and several successive charges. It is important to use certain line systems for washing only, and the valve connections are so arranged that it is awkward to use other than certain line systems and pumps for washing. Leaking valves can also produce a disastrous effect. Every valve in the plant is inspected according to schedule, at least once in every four months, and more frequently whenever a leakage is suspected.

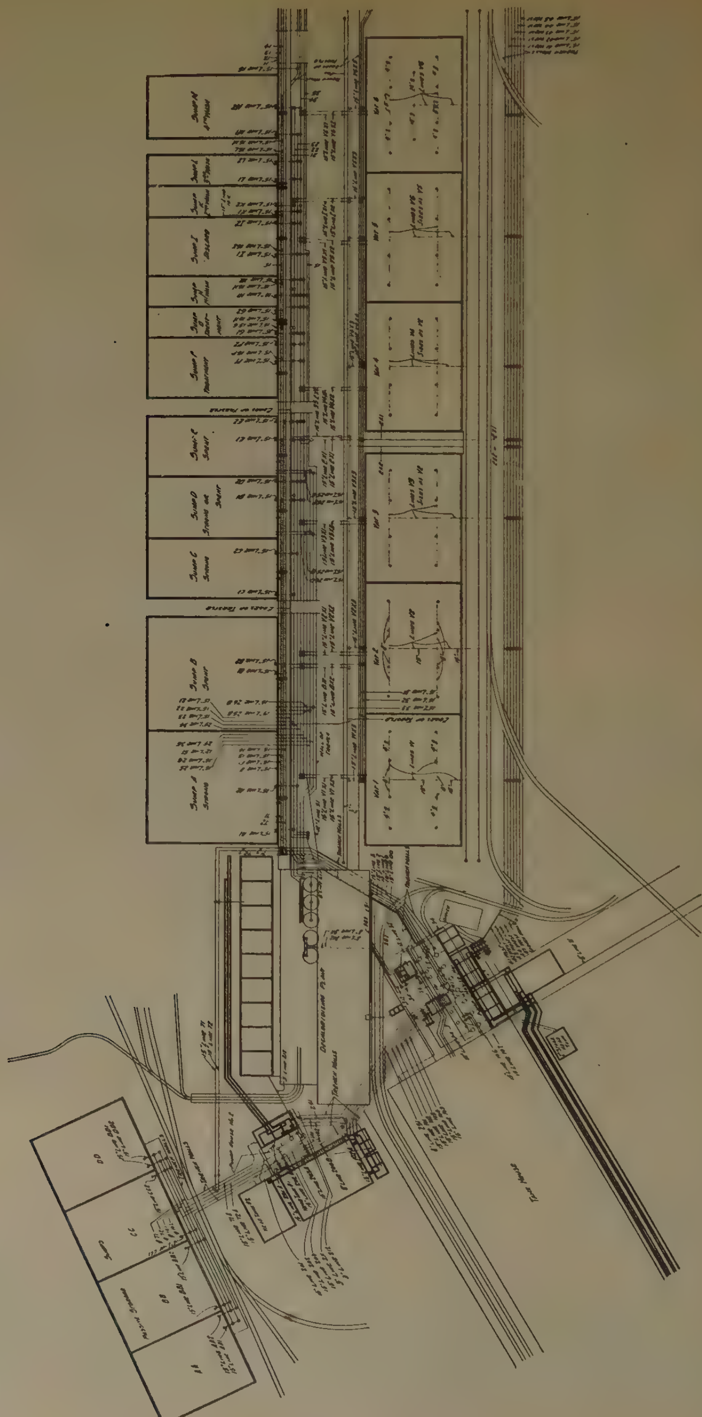
Solution Systems in Detail

There are two sets of wash solutions, one for vats 1 to 6 and the other for vats 7 to 13. There is some irregularity in the volume of the individual washes due to the size of the sumps, and the volume of wash for vats 7 to 13 is larger, in proportion to the capacity of the vats, than the volume for vats 1 to 6. The fourth wash sump for both series is larger than the others, to take care of unexpected fluctuations in the volume of drains. The actual volumes of wash solutions are shown in Table 4.

TABLE 4.—*Volumes of Wash Solutions*

Wash	Vats 1 to 6		Vats 7 to 13	
	Sump	Wash Used, Cu. M.	Sump	Wash Used, Cu. M.
1	H	1450	P	1750
2	K	1450	R	1750
3	L	1450	O	1750
4	M	1850	N	1750
Total..................		6200		7000

(See Fig. 14.) This table and figure show that the average total volume of wash solution used is 6600 cu. m., or about 645 liters per metric ton of ore. The average volume of each wash is 1650 cu. m., the value that was used in general calculations.



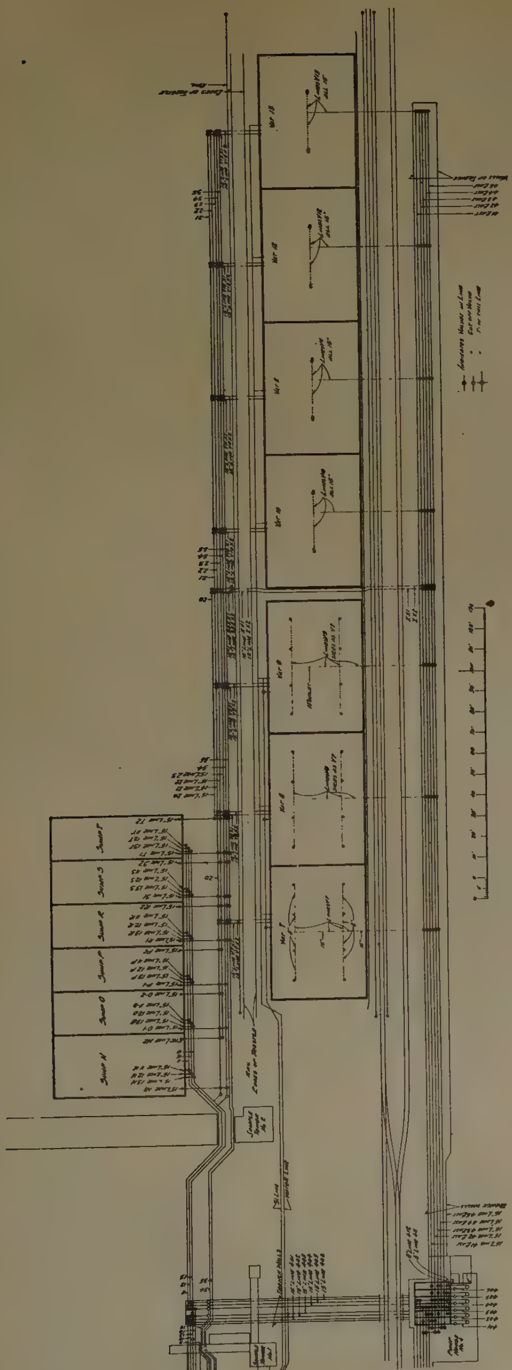


FIG. 14.—LAYOUT OF ACID LINES.

The water is supplied to both sets of vats by a single line from a large steel tank, which is supplied by gravity from the main reservoir, or by reclamation water pumped from the smelter. The feed from the steel tank to the vats is by gravity. The amount of water used per charge has been discussed. About 1700 cu. m. per charge, or 166 liters per metric ton, is the average contemplated. About 200 cu. m. per charge more of water is used on the large vats than the small ones.

The post-treatment discard from both sets of vats is pumped to sump I, which has a capacity of 3200 cu. m. This serves as a small buffer in the production and disposal of solution for washing. The discard solution goes from sump I by gravity directly, or via the tank-house plating-down sections to pump house No. 1 and thence to the decopperizing section of the dechloridizing plant. Spent electrolyte for discard is by-passed from the return to the leaching plant, to the dechloridizing plant directly, or via the tank-house plating-down sections to the dechloridizing plant.

There are two sets of treatment solution, one for each set of vats. Sumps F and G form the storage space for treatment to and from vats 1 to 6, while sumps S and T serve in the same capacity for vats 7 to 13. Either set of sumps can hold the maximum amounts of treatment and advance (which is pumped to the same sumps as the treatment) that are produced from a single charge. Since this solution is usually used as produced, there is perhaps no great necessity for the sumps, but they prove of value in increasing the flexibility of the leaching operation, and are necessary if a vat or sump must be drained unexpectedly. Such a necessity is usually caused by the breaking of an outlet nipple. A breakdown of the unloading bridges may necessitate the use of the treatment sumps to full capacity.

Strong solution from both sets of vats is pumped via pump house No. 4 to sumps A, C, D or E, and from there is transferred to passive storage sumps AA, BB, CC and DD, or goes directly to the dechloridizing plant by gravity. Strong solution can go from the passive storage sumps directly to the dechloridizing plant. Spent electrolyte is returned from the tank house, via pump house No. 2, to sumps B, D or E, or to the passive storage sumps. From all of the sumps mentioned it goes by gravity to any of the vats.

It was originally intended to use the passive storage sumps either for spent electrolyte or strong solution that had been dechloridized. In the latter case, strong solution from the dechloridizing plant could be pumped from pump house No. 2 to the tank-house head tank or, if in excess of the tank house demand, it could be pumped to the passive storage sumps. In case of a shortage of dechloridized strong solution, it could be obtained from the passive storage sumps and replaced there by spent electrolyte.

The decomposition of the nitric acid progressed more rapidly in dechloridized strong solution than in any other solution; therefore it was inadvisable to store this solution whenever the tendency for reaction between the iron nitric acid, and molybdenum was abnormal. It appears that there is some tendency for this reaction to take place in any of the primary solutions, whenever stored in a sump for too long a period. However, there is less danger in storing undechloridized strong solution than dechloridized strong, since the ferric iron produced in the former can be reduced readily in the dechloridizing process. Spent electrolyte can be stored with the least amount of danger, since a passage of the solutions through the ore seems to tend to scrub out some of the liberated NO. Fuming of the solutions, when the decomposition of nitric acid is increasing, is always first noticeable in the treatment solution which, as has been discussed, is the spent electrolyte that has been in contact with the ore for the longest period. As a matter of fact, several dangerous situations have been relieved by circulating the entire outflow from the tank house over leached ore. The relief comes from the cooling effect of the ore, and the tendency for the NO to be expelled from the solution by the scrubbing effect of the ore and the agitation through rapid circulation of the solutions. It is advisable to keep all solutions in movement as far as possible at all times.

Since strong solution and spent electrolyte are potentially the same, the relative proportions of each in the primary system represent what the relation of the supply of copper from the mine and the load on the tank house has been. The purpose of the storage sumps is to form a buffer in the plant process.

To obtain the maximum storage, it is possible to use the same sumps for either strong solution or spent electrolyte, leaving enough space to permit the removal of one class of solution from a sump before the necessity of introducing the other. Mixing strong solution with spent electrolyte is desirable only in the tank-house head tank. Sumps A and C are never used for spent electrolyte, or sump B for strong solution, for the reason previously mentioned and to facilitate the measurement and mixing to uniform grade of the strong solution.

Treatment solution is intermediate between strong solution and spent electrolyte, and is kept on the ore as much as possible. Treatment solution does not go directly from the charge from which it was produced to the next charge treated thereafter, as do the wash solutions. It usually goes from one charge to the fourth or fifth charge thereafter treated, in the same set of vats. It has been mentioned that each set of vats has an individual set of treatment solutions. The number of treatment solutions (or the number of charges that the treatment solution can cover simultaneously) for each set of vats is so arranged that as many charges as possible can be soaking at the same time. To obtain

the maximum, the number of treatments should be such that treatment solution is produced at just about the time it is required to cover a fresh charge of ore. As there is a variation in the intervals of loading of successive charges, it is not possible to make the time of washing correspond at all times to the time of loading, and hence to carry the absolute maximum number of treatments provides no margin of safety, which is required in the light of possible breakdowns of the loading or unloading equipment, or the unexpected necessity of draining a vat. The absolute maximum of treatments in each set is one less than the number of vats in that set, which are in operating condition. However, this requires the full use of the treatment sumps during certain intervals, so that the normal practice is to carry a number of treatments in each set equal to the number of vats minus two. Since, under present operating methods, the spent electrolyte and treatment are almost of identical content, the order of usage of these in the treatment of a charge is not important.

Hence, whenever the mine shuts down over Sunday, it is customary to cover an extra charge in each set of vats with spent electrolyte, thus temporarily increasing the number of potential treatments. An entire treatment is subsequently used as advance in order to get rid of this potential treatment when necessary. Procedure along these lines is often resorted to, in order to tide over emergencies, when it is necessary to take care of an excess or make up a deficiency in the amount of spent electrolyte in the primary system.

The total maximum volume of solution that can be carried with safety in the primary system is shown in Table 5. This contemplates 13 vats in service, and that, when all possible sumps are full of strong solution there shall be no spent electrolyte for storage, or vice versa.

TABLE 5.—*Volume of Solution in Primary System*

Sump	All Strong Solution, No Spent Electrolyte, Cu. M.	All Spent Electrolyte, No Strong Solution, Cu. M.
A	6,400	0
B	0	6,400
C	3,200	0
D	3,200	3,200
E	3,200	3,200
AA	3,800	3,800
BB	3,800	3,800
CC	3,800	3,800
DD	0	3,200
	27,400	27,400

A certain margin of safety is indicated by the content of sump DD, which can hold 3800 cu. m. To facilitate the changing of solutions needed by a change in conditions, and to take care of unexpected volumetric

differences and delays in the discarding of solutions, it is better to limit the total volume of strong solution and spent electrolyte to 26,000 cu. m. Sump DD is equipped with a spray system (for cooling spent electrolyte returned to the tank-house head tank for mixing), as are also the tank-house head tanks. When these are in operation, sump DD is not available for solution storage and the primary system should be diminished by about 4,000 cu. m. under such conditions.

Table 6.—*Maximum Primary Solution*

	Cubic Meters
Tank house and head tanks (at present 9500)	12,000
Spent electrolyte and strong solution (no spraying)	26,000
Treatments for vats 1 to 6 (all in operation) 3200 × 4	12,800
Treatments for vats 7 to 13 (all in operation) 3600 × 5	16,000
Total	66,800

The Cycle

The total length of the cycle, or time from loading to loading of each vat, depends on the rate of production, but certain divisions of this time are of fixed duration regardless of this rate. The average time of charging a vat with ore is and will be about 8 hr. By charging one vat with both units of the crushing plant, this time can be cut down considerably. However, simultaneous loading of one vat in each set is contemplated, and the average time of loading will be about 8 hr. The maximum capacity of the crushing plant is expected to be that of four vats per day.

The covering of the ore with solution will be accomplished during the loading period, so that as little time as possible is lost in the soak, due to loading. However, the operations of loading and covering cannot always be accomplished simultaneously, so that an average delay of 4 hr. should be charged to loading.

The unloading of a vat can be accomplished in 6½ hr. However, the total leaching time lost per cycle through unloading will be 8 hr., as the loading will follow the unloading at this interval.

The washing and draining of a charge can be accomplished in 14 hr., although 16 hr. is usually allowed for this. By the use of two sets of pumps and lines simultaneously, this operation can be accomplished in 10 hr., but it is more often the practice to slow this operation rather than to speed it up. A slower wash gives better results than a fast one, but lengthening the washing time decreases the soaking time, so that a balance must be struck between the effect on the water-soluble and non-water-soluble copper in the tailings in deciding the proportions of the cycle allotted for soaking and washing. At full production, it will undoubtedly be most economical to wash at full pump capacity.

The soaking time and divisions thereof have been fully discussed. The length of this naturally depends upon the length of the cycle minus the time lost through washing, unloading and loading.

Considering absolute maximum capacity as four vats per day for 13 out of every 14 days, and that all 13 vats are in service, the average time from the loading to the loading of each vat will be 84 hours. A division of this time based on the present method of operations is outlined in Table 7. Average times are given.

Items 1, 3, 5, 7, 9 and 10 are more or less fixed, regardless of the cycle. The other items vary with the length of the cycle, all in more or less the same proportion. The number and volume of the strongs will, of course, cause some differences, and item 9 may be lengthened if the length of total soak permits. For best extraction, item 2 should be short and item 8 long. As mentioned, the sum of these two will always form a more or less fixed portion of the total soak, one being lengthened at the expense of the other, and the length of either depending upon the storage space available for strong solution, or upon whether or not the supply from the crusher has exceeded the demand from the tank house for a long period. The relative length of the first and third soaks directly affects the average acid and copper in contact with the solutions during the soak. The calculations and curves covering this subject should be referred to on this point. The approximation of item 1 to the 8-hr. minimum depends on the number of treatments in the system and the coordination of the start of washing on one vat to the loading time of some other.

TABLE 7.—*Loading Cycle*

	Hours
1. Loading and covering (avg. 6 hr. contact time)	10.0
2. Finish of covering or loading to start of first strong	15.0
3. Production of first strong	2.4
4. Finish of first strong to start of second strong	6.4
5 Production of second strong	2.4
6. Finish of second strong to start of third strong	6.4
7. Production of third strong	2.4
8. Finish of third strong to start of washing	15.0
9. Washing and draining	16.0
10. Unloading and delay to start of loading	8.0
Total cycle	84.0

Under the definitions previously given, the following effective soaks would obtain:

Soak	Hours
1. $= 15 + \frac{1}{2} \times 6$ hr.	18
2. = start first to finish third strong	20
3. $= 15 + 2$ hr.	17
Total soak	55

As far as the handling of solutions is concerned, the absolute maximum of production is limited by the interval between the start of washing one charge in a set of vats to the start of washing the next in that set. This can be cut down to 8 hr. by the use of two pumps on the operation during part of the time. Thus six charges per day could be treated. The results would suffer accordingly.

Bluestone Plant

Occasionally, first strong solution is returned to some charge of ore for re-enrichment for the purpose of producing bluestone. This solution is put on fresh ore and the charge dropped out for one complete cycle of the vats, to allow as complete a saturation of the solution as possible. This dropping out is not absolutely essential, but assures a satisfactory batch of solution. When saturation of the solution is reasonably complete, or as soon thereafter as is convenient, enough of this solution to fill one of the bluestone pans is withdrawn from the charge and the leaching thereof completed in the customary manner. There are two bluestone pans, the capacities of which are 540 cu. m. and 425 cu. m. The pans can be safely filled to a depth of 77 cm. and have areas of 700 sq. m. and 550 sq. m. respectively. The bluestone is produced by solar evaporation to 50 per cent. of the original volume in these pans and is accomplished at an average rate of about ¾ cm. per day. The copper sulfate so formed averages about 23.5 per cent. copper, and is free enough from the harmful constituents in the solution to be satisfactory for introduction in the starting sheet solution used in the soluble-anode section of the tank house. It is usually so employed, the purpose being to help in maintaining the desired content in this solution and permit of the purification by discard thereof. A considerable quantity of this bluestone has been sold as a commercial product. The capacity of the plant is approximately 60 metric tons of bluestone per month.

Dechloridizing Plant

The dechloridizing plant was designed primarily to remove the chlorine from the strong solution before electrolysis of the latter. The functions of this plant have been extended to decopperizing the solutions to be discarded and reducing the ferric iron in the strong solution.

As the strong solution enters the dechloridizing plant, a small portion is by-passed to an agitator tank, where it is used to emulsify what is termed washed cement copper. The emulsion formed is fed back into the canal carrying the strong solution, and the solution then passes through a Parral agitator, where the following reactions take place:

$$\underset{\text{in Washed Cement Copper}}{Cu} + \underset{\text{in Strong Solution}}{CuCl_2} = \underset{\text{Precipitate}}{Cu_2Cl_2} \qquad [1]$$

$$\underset{\text{in Washed Cement Copper}}{Cu} + \underset{\text{in Strong Solution}}{Fe_2(SO_4)_3} = \underset{\text{Remain in Strong Solution}}{2FeSO_4 + CuSO_4} \quad [2]$$

Thus the precipitation of the chlorine and reduction of the ferric iron is accomplished.

From the Parral agitator, the strong solution passes through settling tanks and thence to the tank house. The cuprous chloride settles out in these tanks, from which the solution is decanted off in rotation and the cuprous chloride is removed. A crane serves for the digging and handling of the cuprous chloride, which is conveyed to mixing boxes. In the mixing boxes, the cuprous chloride is brought in contact with warm $FeCl_2$ solution, called brine. It is dissolved in this brine, possibly according to the following reaction:

$$\underset{\text{Brine}}{FeCl_2} + \underset{\text{Cuprous Chloride}}{Cu_2Cl_2} = \underset{\text{Enriched Brine}}{Fe(CuCl_2)_2} \quad [3]$$

The enriched brine solution passes from the mixing boxes to precipitation drums, which are loaded with scrap iron. In these drums cement copper is precipitated according to the following reaction:

$$\underset{\text{Enriched Brine}}{Fe(CuCl_2)_2 + Fe} = \underset{\text{in Emulsion from the Drums}}{2FeCl_2 + 2Cu} \quad [4]$$

The emulsion of cement copper in brine solution passes from the drums through small classifying tanks, wherein the coarsest of the cement copper grains are settled out. The product from these tanks is removed periodically and prepared for shipment to the smelter by drying; sometimes washing and drying. From the classifying settlers, the emulsion passes through other tanks wherein the balance of the cement copper settles out. This product is called unwashed cement copper. The brine passes through the last mentioned settling tanks to a head tank, from which it is pumped by an air-lift back to the mixing boxes for use in dissolving more cuprous chloride. It can be noted that the $FeCl_2$ content of the brine is increased with each contact with the cuprous chloride. To keep the concentration of this from rising, water is added to the brine, and the excess volume of brine so produced is bled or discarded from the system to the wash agitator used in the decopperizing process. An attempt is made to keep the brine solution at from 26° to 30° Bé. The reaction in the drums furnishes heat, which makes the brine more efficient in dissolving Cu_2Cl_2. To keep the content of the brine solution at a balance, as much $FeCl_2$ must be bled from the system as is put in, or all the chlorine removed from the strong solution must eventually go in the brine solution bled off, into the decopperizing process.

The unwashed cement copper is picked up by a crane and carried to the wash agitator into which the discard of brine solution was passed. In this agitator the brine and cement copper come in contact with the

post-treatment solution or the spent electrolyte that is to be discarded. The copper is precipitated from the last mentioned solutions according to the following reaction:

$$\underset{\text{Unwashed Cement Copper}}{2Cu} + \underset{\text{Brine Discard}}{2FeCl_2} + \underset{\text{Post-T or S. E. Discard}}{CuSO_4} = \underset{\text{Washed Cement Copper}}{Cu_2Cl_2 + Cu} + \underset{\text{In Solution to be Discarded}}{FeSO_4 + FeCl_2} \quad [5]$$

The solution to be discarded passes from the wash agitator to other tanks, wherein the washed cement copper settles out. The discarded solution passes through these settlers and from there to the so-called pampa tanks. Since the settling is not perfect nor is the balance of reaction 5, the settling and precipitation of the copper from this solution is further completed in these pampa tanks. Into these tanks scrap iron is charged. The outflow from the dechloridizing plant settlers averages about 0.5 g.p.l. copper and from the pampa tanks about 0.3 g.p.l. copper. From the latter tanks the solution is wasted.

In reaction 5 an extra atom of copper is indicated, which does not enter into the reaction. An excess of copper is used in this process so that the product (washed cement copper) is suitable for use in dechloridizing the strong solution. This wash cement copper is removed from the settlers and carried to the agitator tank in which the dechloridizing emulsion is formed. The copper in this product is that entering into reactions 1 and 2. It will be noted that the Cu_2Cl_2 in this product was not noted in these reactions. It takes no part therein and is simply carried in the strong solution with the Cu_2Cl_2 precipitated therefrom, to the cuprous chloride settlers.

An excess molecule of $FeCl_2$ was also indicated in reaction 5. This simply denotes that the $FeCl_2$ picked up in the processes must eventually all go to the pampa. It would appear at first that part of the chlorine from the brine discard was retained. This retention is only apparent, as the Cu_2Cl_2 formed in the decopperizing process augments that formed from the dechloridizing of the strong solution and hence the amount that enters the brine solution and the amount in the brine discard, as previously explained.

As a matter of fact, the amount of Cu and Cu_2Cl_2 shown in any of the reactions and products are not exactly as indicated. There is usually an excess of both constituents in all the reactions, and the products cuprous chloride, wash cement copper and unwashed cement copper are all mixtures of Cu and Cu_2Cl_2 in various proportions, although unwashed cement copper carries very little Cu_2Cl_2. These products are also saturated with the solutions with which they have been in contact, and carry moisture values of Cu, Cl and Fe accordingly. It is a part of the purpose of the decopperizing process to remove the $FeCl_2$ moisture content of the

cement copper used in dechloridizing. Formerly unwashed cement copper was used in dechloridizing. This was, of course, saturated with brine solution and introduced iron into the strong solution, which was undesirable. In the decopperizing process, this brine is diluted and replaced by the solution for discard which is in great excess of the brine solution entering the process. For this reason, the decopperizing process is frequently called the washing process. Even with this washing, the product, washed cement copper, carries some iron which is introduced into the strong solution. To reduce this amount, a second washing of the cement copper is sometimes employed. To accomplish this, the product from the decopperizing process is carried to an agitator wherein it comes in contact with the discard solution alone. From this agitator the solution passes through settlers, where the doubly washed copper is settled out. The solution which has not been materially affected by the operation is pumped back to the agitator, where it comes in contact with the brine discard and the unwashed cement copper, and is decopperized according to reaction 5. The doubly washed cement copper, which is the same as the washed except for some modification of the moisture content, is the product used in dechloridizing the strong solution. So far, little benefit has been apparent from the application of this double washing, and the settling area for solutions to be wasted is cut down thereby, sufficient to noticeably increase the copper content of the solutions wasted. At full production, it would be impossible to use this double wash without considerable loss of copper. Since with the single wash the amount of iron introduced from the washed cement copper appears to average only 13 g. per ton of ore treated as against 220 g. from the ore, the double washing seems scarcely justified.

It has been suggested that the washed cement copper grains are so coated over by Cu_2Cl_2 that the efficiency of this product in dechloridizing is greatly reduced. This is undoubtedly often true, but the ratio of the discard to brine solution and unwashed copper used in decopperizing produces considerable variation in the composition of the washed cement copper. These variations are not always simple of control. A scheme was tried which consisted of washing part of the unwashed cement copper in the discard solution without the addition of brine and using this product for dechloridizing, with the remainder of the unwashed cement copper going into the decopperizing process and returning the product of this process to the unwashed copper-settling tanks. In these tanks the Cu_2Cl_2 would be dissolved from the copper grains by the brine solution, and the product re-used as unwashed cement copper. The trouble with this scheme is that the unwashed cement copper carried enough $FeCl_2$ moisture content to partially complete the decopperization of the discard solution, without the addition of the direct brine discard, and hence became fouled to a certain extent by Cu_2Cl_2.

Analyses of the products in the dechloridizing plant are shown in the results that will subsequently be tabulated. The proportions of Cu and Cu_2Cl_2 in each can be judged from these.

The only source of chlorine to the dechloridizing plant is the strong solution. Should the chlorine content of the ore disappear, it will be necessary to add salt to the ore or modify the process. The same chlorine is apparently used over and over again, and a stock of Cu_2Cl_2 once formed could theoretically be maintained, without a source of chlorine, but it is impossible under operating conditions to do this, since chlorine is continually lost at certain points.

The copper content of the waste solution from the decopperizing process increases more rapidly with a deficiency of Cl to satisfy the Cu than with an excess, although the Cu_2Cl_2 in the product is soluble in $FeCl_2$. Therefore, a slight excess of $FeCl_2$ is usually maintained in the waste solution.

There is a certain amount of chlorine in the cement copper removed from the system and sent to the smelter, even when this product is especially treated for the removal of chlorine.

The settling is not perfect and a certain amount of cuprous chloride is carried in suspension into the tank house. A portion of this finally appears as tank cleanings and never gets back to the dechloridizing plant.

It is true, however, that with careful management, the process may be maintained with a low chlorine content in the ore, although when this content drops to the point where a normal discard of solutions can maintain the concentration of chlorine in the primary system at 0.15 g.p.l. it will not be economical to continue the present dechloridizing or decopperizing processes for these purposes only. Below 0.15 g.p.l. the chlorine in the electrolyte will not be harmful enough to necessitate removal. When the chlorine content of the ore drops to about 0.01 per cent., this condition will probably be reached, although the acid gain from the ore will have to be considered. The point of discard will have to be varied to keep the desired acid concentration in the primary system, and since without dechloridizing, the concentration of chlorine and total available acid will follow one another in the solution, the ratio of the acid gain to the chlorine content of the ore must be considered rather than the chlorine content alone. The figure 0.01 is lower than that usually given in this connection, but the reason for this is that a drop in the acid-making constituents of the ore along with a drop in the chlorine content is contemplated.

Bearing in mind that one atom only of the copper precipitated from Cu_2Cl_2 by Fe is net, in the present process, the cost of electrolytic deposition of copper is far less than that of precipitation by scrap iron, except where the former is carried to a very low copper content of the cell outflows, although it must be considered that the dechloridizing plant ever

since its inception has borne considerable expense of experimentation and modification of equipment. For this reason, the fact has been brought out that it will undoubtedly be economical to discontinue dechloridizing when the ore content drops to 0.01 per cent., if not sooner, and decopperize the discard solutions by electrolytic deposition to as low a point as possible without producing soft cathodes, and terminate the decopperization by scrap iron. Under such conditions the iron in the primary solution must be taken care of by some adaptation of neutralization or partial neutralization and limestone purification of the strong solution, or reduction by SO_2, which might be feasible should the increase in sulfide copper in the ore sufficiently accompany the drop in chlorine and acid-bearing constituents. It is also probable that a drop in the nitric acid content of the ore will accompany the drop in chlorine, and if the nitric acid is the only constituent of the solution directly affected by the molybdenum, the normal solution discard will probably keep the iron content of the solution within bounds and the trouble from ferric iron will not be excessive. This control of the iron content of the solutions by discard will also depend upon the ratio of the soluble iron and acid-making constituents of the ore. The former should also decrease in the future, although recently the iron gained from the ore has been abnormally high and the iron picked up from the ore has been ferric iron.

The sources of copper entering the products in the dechloridizing plant are from the Cu_2Cl_2 in the strong solution and the copper content of the discard solution. Of this copper a certain percentage (approximately 8 per cent.) is carried out of the plant in suspension in the strong solution to the tank house and the waste solution to the pampa tanks. The remainder either goes into the strong solution in reducing ferric iron or to the smelter, where it is converted into soluble anodes. The latter are used in the tank house in the production of starting sheets. By either route the copper eventually goes into the tank-house cathodes, and at about the same total expense. Since good starting sheets can be produced from the primary solution in soluble-anode cells, and this is proving practical in the tank house at present, there is no purpose in producing cement copper for the smelter, and no additional expense is caused through the reduction of the ferric iron.

The amount of copper removed from the strong solution is inevitable, depending on the chlorine content thereof. The amount removed from the solution for discard depends upon the copper content to which this solution is plated down before its treatment in the dechloridizing process. When plating down has been carried to an economic limit (remembering the benefit of washing the iron and chlorine from the cement copper used in dechloridizing) the copper removed from the processes is available for the reduction of ferric iron, and the expense of precipitation of the cement copper so used from Cu_2Cl_2 should not be charged to this reduction.

However, if the content of the discard solutions is raised to meet the requirements of reducing ferric iron, the precipitation of the increased copper content of the discard solution should be charged to the reduction of ferric iron.

In the scrap iron precipitation of cement copper, about 7 per cent. of the product is too coarse for efficient use in the processes. This portion consists of copper grains over 200 mesh in size. It is advisable to remove sufficient of the drum product to insure the removal of the coarse grains from the system. If about 10 per cent. of the copper gained in the plant is removed in the classifiers, previously mentioned, the results will be satisfactory; thus, 10 per cent. for coarse copper and 8 per cent. for loss in product carried out in suspension by the solutions should be deducted from the copper removed in the process, in determining the amount available for the reduction of ferric iron.

Since the direct purpose of precipitating copper from Cu_2Cl_2 is not to form cement copper for specific uses, but to decompose the Cu_2Cl_2 inevitably formed, the scrap iron and drum capacity required should be calculated with regard to the minimum amount of Cu_2Cl_2 that must be decomposed. This, as previously shown, depends directly upon the copper in the discard solution and the chlorine in the ore. Bearing in mind that it is necessary to precipitate an extra atom of copper for each one removed from the solution, the amount of copper that must be precipitated can be figured from the following equation:

$$C = 2 \times L \times D + 17{,}941 \times N$$

Where,

C = grams cement copper necessary to be precipitated per metric ton ore treated.
L = liters solution discarded per metric ton ore treated.
D = grams per liter copper in solution for discard (after plating down when employed).
N = percentage of chlorine in the ore.

From this result for C, it is usually safe to subtract 10 per cent. for Cu and Cu_2Cl_2 carried out in suspension and for incomplete decopperization of the discard solution, and incomplete dechloridization of the strong solution. This percentage will increase with lower content of the chlorine in the ore and copper in the discard solution. It may also be modified by a change in the settling conditions, or in the method of handling the product from the pampa tanks and the head tank cleanings.

Under the possible existing conditions of operation, the amount of scrap iron consumed in precipitating copper from Cu_2Cl_2 depends mainly upon the class of iron used. Wrought iron and steel seem to give the best results, while castings and so-called bailed tin cut down the efficiency

considerably. Over the last four years the scrap iron consumed has averaged approximately 0.6 times the copper precipitated. In Tables 8 to 10 it is higher than this, but a large percentage of the so-called bailed tin was used during the period represented.

The cement copper removed from the classifiers is a wet sloppy product with a high iron and chlorine content, due to the brine moisture that it carries. This product is placed in collecting tanks until one of the preparation tanks is available; then it is transferred to the latter. The preparation tanks are equipped with filter bottoms and the moisture content of the product can be reduced to from 10 to 15 per cent. therein. This product is cheaper to handle and treat in the blast furnace than it would be in the original condition. The iron content of the product is at times desirable for the blast furnace but the chlorine content is not; therefore it is sometimes the practice to wash this product as well as dry it before shipment to the smelter. For the purpose of washing, the product is covered with the waste solution from the decopperizing process. This solution displaces most of the brine moisture and is considerably lower in chlorine and iron than the latter. The waste solution is then displaced from the product by water, which is drained off through the filter bottom. If water alone is used for washing, basic iron salts are thrown down and the iron will not be removed. The waste solutions are high in acid and the precipitation of the basic iron in the product is prevented by the application of the solution as described above.

TABLE 8.—*Statistics of Operation from Oct.* 1, 1927 *to Apr.* 1, 1928

Number of charges treated	322
Metric tons of ore treated; average per charge, 10,176; total	3,276,715
Total copper in ore treated, per cent	1.622
H_2SO_4 insoluble copper in ore treated, per cent	0.053
Oversize on 0.371-in. mesh in ore treated, per cent	9.181
Total copper in tailings, per cent	0.167
H_2O soluble copper in tailings, per cent	0.022
H_2O in tailings, per cent	8.271
Extraction of copper, per cent	90.242
Acid gained per metric ton ore treated, kilograms	7.166
Average duration of soaks, hours	
First	27.19
Second	41.29
Third	22.25
Total	90.73
Average net washing time, hours	13.19
Average draining, hours	3.51

Average content of solutions in contact with ore during soak, 29.97 g.p.l. copper and 55.24 g.p.l. free acid.

Average content of covering solution before washing, 17.41 g.p.l. copper and 73.84 g.p.l. free acid.

Scrap iron consumed per 100 kg. copper precipitated, 65.86 kg.

In the manner of washing and drying described, a cement copper product has been prepared that can often be charged directly into the anode furnace without previous preparation in the blast furnace.

Tables 8 to 10 show results from the operation of the leaching and dechloridizing plants from Oct. 1, 1927 to Apr. 1, 1928.

TABLE 9.—*Quantities and Contents, Oct. 1, 1927 to Apr. 1, 1928*

Item	Total Cubic Meters or Metric Tons[a]	Average per Metric Ton Ore Treated, Liters or Grams[b]	Cu, g.p.l.	H_2SO_4, g.p.l.	Cl, g.p.l.	Total Fe, g.p.l.	Ferric Iron, g.p.l.
Solution used per charge in original cover of ore	3,336m	3,281	16.13	65.58			
Strong solution produced per charge	6,670m	6,551	35.41	47.77			
Solution used displacing strong per charge	6,670m	6,521	14.19	75.10			
Treatment solution produced per charge	3,336m	3,281	15.21	64.50			
Advance and solution to decopperizing produced per charge	996m	981	8.83	37.69			
Total wash solution used per charge	6,478m	6,391					
First wash solution			7.28				
Second wash solution			4.15				
Third wash solution			2.71				
Fourth wash solution			1.85				
Water used per charge	1,888m	1,861					
Strong solution from leach plant to dechloridizing plant	2,136,860m	6,551	35.39	47.85	0.59	4.60	1.37
Strong solution from dechloridizing plant to tank house	2,136,860m	6,551	36.19	46.09	0.13	4.62	0.01
Chlorine removed	985t	301g					
Ferric iron reduced	2,912t	889g					
Spent electrolytic returned tank house to leaching plant	2,000,763m	6,111	14.44	78.54	0.21	4.60	1.13
Spent electrolyte to decopperizing	55,697m	171	14.89	74.13			
Plated down S. E. to decopperizing	66,977m	201	9.10	86.85			
Post-treatment to decopperizing	152,377m	471	9.77	36.67			
Total solution to decopperizing	275,051m	841					
Copper in solutions to decopperizing	2,965t	905g					
Total available acid in solution discard	19,622t	5,988g					
Waste solution from decopperizing process	298,451m	911	0.56				
Same from pampa tanks	298,451m	911	0.55				
Copper from pampa tanks	107t	33g					
Copper to smelter from dechloridizing plant	1,359t	415g					
Copper to smelter from pampa tanks	45t	14g					
Copper precipitated from Cu_2Cl_2	6,796t	2,075g					
Scrap iron consumed in precipitating copper	4,475t	1,366g					
Bluestone made 23.58 per cent. Cu	268t	82g					

[a] m denotes meters; t, tons. [b] l denotes liters; g, grams.

TABLE 10.—*Average Analyses*

PRODUCTS IN DECHLORIDIZING PLANT, OCT. 1, 1927 TO MAR. 1, 1928

Item	H_2O, Per Cent.	Cu, Per Cent.	Fe, Per Cent.	Cl, Per Cent.
Cement copper to smelter from dechloridizing plant	14.8	80.0	4.6	4.2
Cement copper to smelter from pampa tanks	25.0	43.0	7.2	1.4
Unwashed cement copper in process		72.4	7.5	9.2
Washed cement copper in process		72.6	1.2	12.4
Double washed cement copper in process		72.8	0.7	14.9
Cuprous chloride in process		68.3	0.4	18.6

COMPOSITE SAMPLES OF SPENT ELECTROLYTE, OCT. 1, 1927 TO APR. 1, 1928

	Grams per Liter		Grams per Liter		Grams per Liter
Cu	14.44	Al	1.75	K	2.30
Acid	78.54	As	0.242	Pb	0.011
Cl	0.21	Sb	0.009	Sn	0.004
Fe	4.60	Ca	0.475	Mn	0.135
SO_4	144.83	Mg	0.147	Mo	0.248
HNO_3	3.12	Na	6.74	Total solids	197.78

LEACHING PLANT EQUIPMENT

The vats and sumps are of reenforced concrete with mastic lining. The lining is 4 in. thick and is also reenforced, the reenforcing being attached to the concrete walls by iron hooks set therein. The present type of mastic lining gives evidence of a life of 20 years with little repair. The capacity of the sumps has been given. All these, with the exception of the passive storage sumps, are 12 ft. deep; the latter are 14½ ft. deep. This depth is inclusive of the mastic on the floor. Vats 1 to 6 are 150 by 110 by 17 ft. 5 in. deep. Vats 7 to 13 are 150 by 110 by 19½ ft. deep. The sumps and vats are equipped with mastic discharge nipples. Each sump has a telltale capacity indicator for measuring the solution therein. The floats for these telltales are simply old glass solution carboys, which travel in wooden guides. The indicator scales are calibrated to read direct in cubic meters.

The vats are equipped with filter bottoms, which are composed of 6 by 6 in. Oregon pine timber, covered by two layers of 2 by 6-in. Oregon pine boards with a layer of cocoa matting between. The 6 by 6-in. timbers are laid in rows 10 in. apart, the ends of the pieces beveled to permit the passage of solution. The first layer of 2 by 6-in. boards is laid at right angles to the timber and the space between the boards is 3 in. The cocoa matting covers this layer. The matting comes in strips 3 ft. wide. The strips are lapped for about 3 in. The top layer of boards is at right angles to the preceding layer. The space between these boards is ¾ in.

The whole is so arranged that the top layer runs parallel to the bite of the digging bucket, which increases the life of the boards considerably. The average life of this top layer of boards is about 4½ years, and there is considerable replacement during that time. The vats are not dug closely, to save the filter bottoms.

Recently a filter bottom of a different type was installed in vat 8. The cocoa matting was placed over a false bottom that extended for an area of 384 sq. ft. over each of two outlets. The cocoa matting was protected by heavy timbers, the arrangement being such that each individual piece could be easily replaced. The rest of the vat floor was covered by odd pieces of timber and boards, simply to protect the mastic. The life of such a filter bottom should be indefinite, with a certain amount of replacement, and it should be possible to dig this vat cleaner than the others and hence give it more capacity. In this vat, 119 yards of cocoa matting are required as against 1833 yards in the others. The results from this vat seem to be as good as in the others, although there is still some doubt as to whether or not the small area of false bottom will eventually choke up.

The size and lengths of the wood-stave pipe lines can be found in Fig. 14. They are manufactured locally. The staves are milled from 2 by 4-in. Oregon pine. The hoop bolts are of wrought iron, of ½ in. and ⅝ in. dia. The bolts are spaced at 6-in. intervals on the pipe. The pipes are made up in 16-ft. lengths, except where shorter lengths are required. The tongue and groove joints of the pipe are filled with mastic paint. The pipes are joined by wood-stave couplers. In connecting a pipe to a fitting, a cast-iron bell flange is placed on the end of the pipe. Formerly, it was customary to line the bell flanges with lead before placing them on the pipes. It has been found better to cast the lining material between the pipe and bell flange. The pipe standing vertically is kept in position in the bell flange by a die. The molten material for the lining is then poured between the pipe and bell flange. This lining material is lead with about 20 per cent. antimony, practically type-metal composition. Recently, some linings have been cast with the lower part lead antimony and the upper 2-in. mastic. These give promise of being satisfactory. The lining is cast so that a protecting head is formed on the face of the bell flange. Lead pipe, mastic pipe, cast-iron lead-lined pipe, redwood pipe and Oregon pine pipe have been tried. The last is by far the most satisfactory. A large portion of the installation is of comparatively recent date, so that no reliable figures are available as yet as to the exact average life of a piece of pipe. So far, it is indicated that this average will be about 10 years, except in the first 200 or 300 ft. of a pump delivery, where the pipes suffer from vibration and irregularity of pressure. The material for the pipe is selected lumber, but about 5 per cent. of this

material is rejected on account of improper seasoning and grain of the wood.

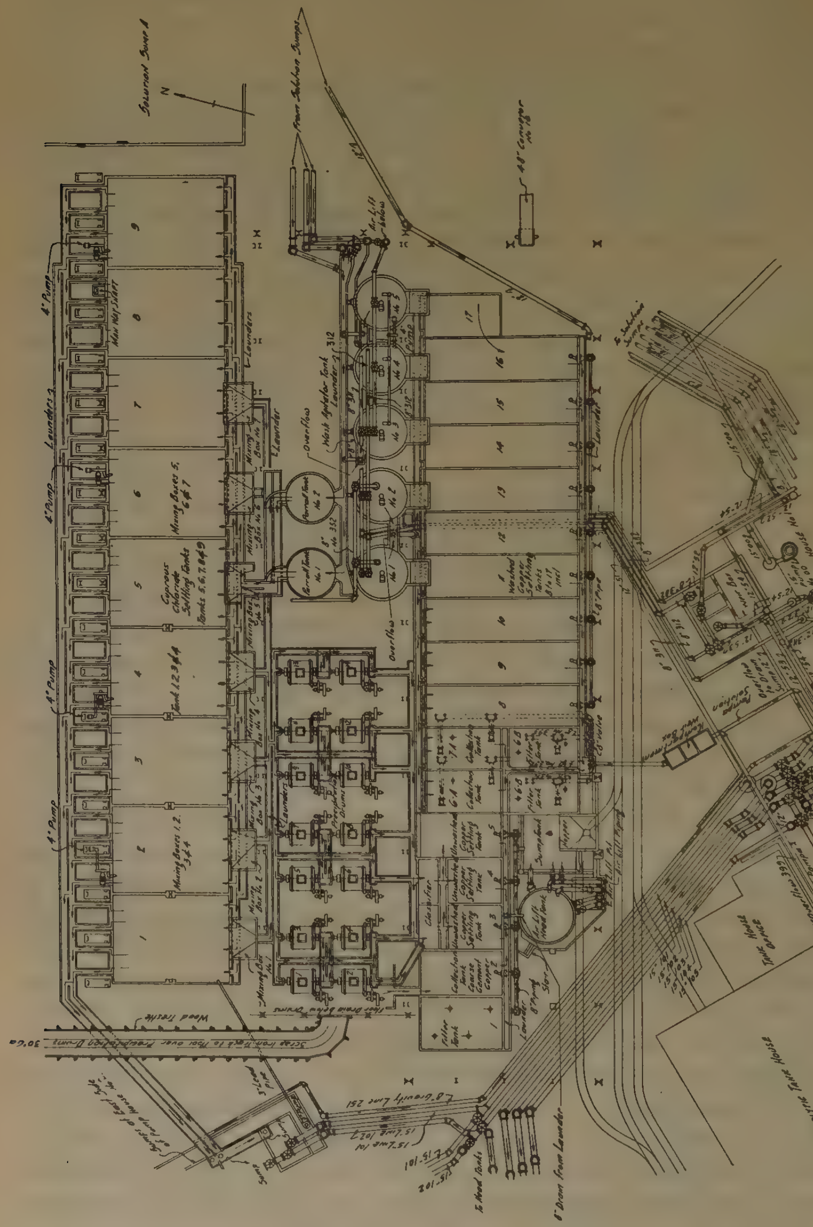


FIG. 15.—GENERAL PLAN OF DECHLORIDIZING PLANT.

The pipe fittings are of cast iron and are lined with mastic. Sulfur linings have been tried; some gave evidence of long life, but when an

attempt was made to reline some of these fittings it was found that the sulfur had attacked the cast iron and ruined the entire fitting. Duriron seats are placed in the valves. The valve spindles are of bronze and covered with lead. The spindles carry a rubber disk inserted in the lead, which seats against the duriron. Some idea of the magnitude of the installation may be gleaned from the fact that there are over 600 valves in the leaching plant lines. Centrifugal pumps are used (Table 11).

TABLE 11.—*Data on Centrifugal Pumps*

Pump No.	Size, Inches	Type	Motor Horsepower	Estimated Average Capacity, Cubic Meters per Hour	Service
00	15	Vert.	250	800	Washing vats 1 to 6.
0	15	Vert.	250	800	Washing vats 1 to 6.
1	15	Vert.	250	800	Spare for 00, 0, 2 and 3, and for pumping undechloridized strong to passive storage.
2	15	Vert.	250	800	Returning spent electrolyte from tank house to leaching plant or to passive storage.
3	15	Vert.	250	800	Same as 2.
4	15	Vert.	250	800	Spare for 5, 6 and 7.
5	15	Vert.	250	800	Returning spent electrolyte to tank-house head tank; can also pump to passive storage.
6	15	Vert.	250	800	Same as 5.
7	15	Vert.	250	800	Same as 5.
11	9	Vert.	75	250	Circulating starting strong solution.
12	9	Vert.	75	250	Circulating starting strong solution.
21	3	Vert.	7½	40	Drain in pump-house floor and picking up spills.
22	3	Vert.	7½	40	Same as 21.
23	4	Horizon.	20	70	Same as 21.
31	9	Vert.	75	250	Pumping discard solution to dechloridizing plant.
32	9	Vert.	75	250	Pumping discard solution to dechloridizing plant.
201	15	Vert.	250	800	Pumping dechloridized solution to tank-house head tank or passive storage.
202	15	Vert.	250	800	Same as 201, and for relaying spent electrolyte or undechloridized strong from pump house 1 to passive storage.
203	15	Vert.	250	800	Same as 202.
211	3	Vert.	7½	800	Picking up spills.
212	3	Vert.	7½	800	Picking up spills.
311	6	Vert.	40	125	For use in second washing of cement copper.
312	9	Horizon.	75	125	Same as 311.
321	4	Horizon.	20	70	Pumping strong for emulsification of cement copper for dechloridizing.
331	4	Horizon.	20	70	Siphoning off cuprous chloride settlers.
332	4	Horizon.	20	70	Same as 331.
333	4	Horizon.	20	70	Same as 331.
334	4	Horizon.	20	70	Same as 331.
401	15	Vert.	250	800	Producing strong solution from all vats.
402	15	Vert.	250	800	Same as 401.
403	15	Vert.	250	800	Spare for 401, 402 and emergency.
404	15	Vert.	250	800	Spare for 405, 406 and emergency.
405	15	Vert.	250	800	Washing vats 7 to 13.
406	15	Vert.	250	800	Washing vats 7 to 13.
411	6	Vert.	40	125	Picking up spills.
412	6	Vert.	40	125	Picking up spills.

Note that sometimes the feed of solution to the pumps is choked down to give a slower wash. Also, the rate of solution handling during draining is also somewhat less than full pump capacity. The operation of producing strong solution is always at full pump capacity. The equipment and connections are so arranged that any operation can be speeded up above maximum pump capacity by using in duplicate two of any set of pumps and lines laid out for the service of this operation.

Pumps numbering less than 100 are in pump house 1; those numbering 200 are in pump house 2; those numbering 300 are in the dechloridizing plant, and those numbering 400 are in pump house 4.

It has been indicated that the line systems were in groups for certain specific operations. A study of Table 10 and Fig. 14 will make this grouping clearer than any detailed explanation. It would be well to mention, however, that the 24-in. lines 34 and 35 are for the specific use of delivering spent electrolyte and advance to all the vats, and are equipped with separate cross-over deliveries to each vat. The 50 lines are for handling solution for discard.

The vertical pumps are all equipped with S.K.F. ball bearings, with the exception of the 3-in. The horizontal are not so equipped. The shafts, casings and stay rods of the vertical pumps are all lined with antimony lead. The impellers are cast of the same material, and contain a cast-iron spider. The pump boots are lined with mastic, and the check valves, where used, are lined with either material. A lead-antimony mixture of about 10 per cent. antimony has proved the most satisfactory. The horizontal pumps have either lead-antimony or duriron casings and impellers. As spare, one pump of each size is kept set up ready for installation in each pump house. According to a schedule, each pump in service is taken out and replaced by the spare, whether trouble is indicated or not. Minor repairs are made to the pump removed, which then becomes the spare. Serious breakdowns of several pumps at the same time are easily avoided in this manner, and there have been very few heavy repair jobs since the adoption of this scheme of revision.

The pump sumps are mostly of concrete with mastic lining, but solid reenforced mastic sumps have been installed in pump house 4.

Each group of vats is served by two unloading bridges. The bridges for vats 1 to 6 carry 6-ton digging buckets. One bridge for vats 7 to 13 carries a 12-ton bucket, and the other an 8-ton bucket. The buckets are of the clamshell type, and discharge into 50-ton hoppers carried by the bridges. From these hoppers the dump cars are fed. Trains of 20 cars each run in duplicate serve each set of bridges. The cars are of the western dump type with side discharge. The tailings disposal tracks are standard gage. Oil-burning locomotives for the tailings disposal have been replaced by electric locomotives and an installation of part third rail and part trolley has been made for this purpose.

Dechloridizing Plant Equipment

A general plan of the dechloridizing plant is shown in Fig. 15, and sections of the layout in Fig. 16. The sumps and wash agitators are of concrete and are lined with mastic. The Parral agitators and brine tanks are of solid reenforced mastic, as are the canals. This type of construction has proved satisfactory under certain conditions.

Wash agitators 3 and 4 serve as emulsifiers for the cement copper used in dechloridizing. Wash agitators 1 and 2 serve in the operation of decopperizing discard solution. Wash agitator 5 serves in the second washing of the cement copper when this operation is employed. The Parral agitators serve in the operation of dechloridizing strong solution. A mechanical agitator has recently been installed to speed up the solution of Cu_2Cl_2 in the brine solution. Formerly, the Cu_2Cl_2 was simply dumped into conical hoppers, called mixing boxes, and the brine solution run through the same. The hoppers had a side discharge near the bottom. The mechanical agitators are driven by 30-hp. motors. The agitator shafts are of tobin bronze and the impellers of bronze. The Parral agitators have proved more satisfactory than the others, as they are much simpler as regards mechanical repair, but in handling the heavier emulsions they give trouble. One agitator in each set is normally sufficient for the service required; the second serves as a spare.

The pump installation has been outlined. In addition to this, air-lifts are used to handle the brine solution, which is both highly corrosive and abrasive. No comparatively cheap material for pump parts or linings has been encountered that will satisfactorily withstand the action of the brine solutions.

Air lifts have proved highly satisfactory for this service; two 8-in. air lifts are used for handling the brine solution, one 4½-in. triple-stage air lift is used for draining the air-lift pit. This pumps from a small pit 2 ft. deep, which is in the bottom of the main air-lift pit. The drain lift will rapidly remove solution to clear the floor of the main air-lift pit, although the lift is 44 ft. Two 8 in. air lifts are installed in the main pit, for returning brine solution from a surge tank to the main air-lift head tank. There is also an 8-in. air lift for duplicating the service of pump 321. The pump is usually kept as a spare for this air lift. The lifts are of the injector type (Fig. 17), bronze injectors containing a high percentage of copper are used.

There are nine tanks for settling the Cu_2Cl_2 from the strong solution. The total settling area of these is 8433 sq. ft. As one of these tanks is usually out of service for digging Cu_2Cl_2, the area will be rather scant at full production. The solution is siphoned off from these tanks by pumps, before digging.

For the settling of unwashed cement copper or the drum product, cement-copper settlers 2, 3, 4 and 5 are available. The total area of these

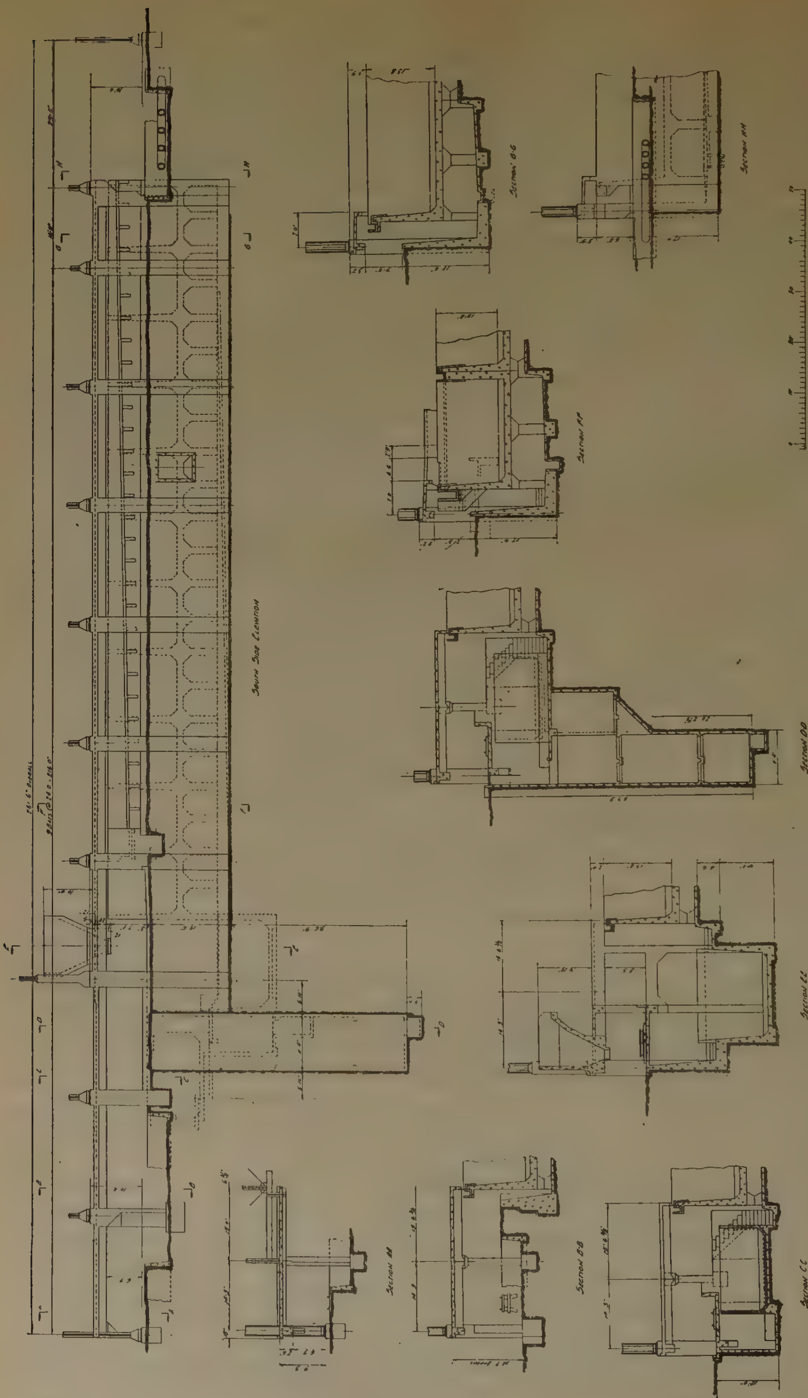
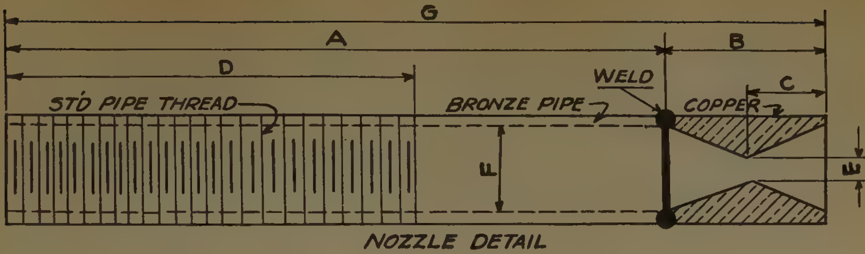
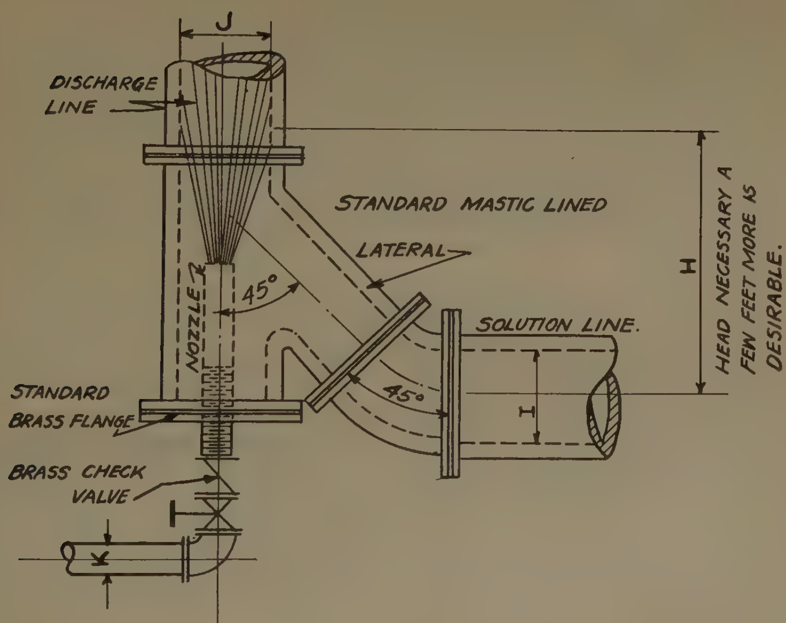


FIG. 16.—LAYOUT OF DECHLORIDIZING PLANT.



Size of Nozzle, In.	A, In.	B, In.	C, In.	D, In.	E, In.	F, In.	G, In.	Solution Line, Size of Pipe, In.
2	18	4	2	11	5/8	2	22	15
1½	16	3½	1¾	10	½	1½	19½	10–12
1	14	3	1½	9	3/8	1	17	8–9
¾	10	2½	1¼	6	¼	¾	12½	4–6
½	8	2	1	5	3/16	½	10	3–4



Solution Line, Size of Pipe, In.	H, In.	I, In.	J, In.	K, In.	Size of Lateral, In.
3 to 4	20	3	3	½	Special lead
4 to 6	24	4–6	6	¾	8 × 8 × 8 with reducer
8 to 9	30	8	8	1	8 × 8 × 8 standard
10 to 12	34	12	12	½	12 × 12 × 12 reducer for 10 in.
15	38	15	15	2	15 × 15 × 15 standard

FIG. 17.—INJECTOR TYPE OF AIR LIFT.

is 1084 sq. ft. One of these is always out for digging. The coarse copper is removed from the emulsion leaving the drums in the two small tanks marked classifiers. These are 6 ft. deep and each has an area of 80 sq. ft. All the drum product passes through these classifiers before entering settlers 2 to 5. The classifiers are not decanted off for digging but are in continuous service. From the classifiers, the coarse copper is dug and placed in settlers 6A and 7A, which simply serve as storage tanks for the cement copper until one of the preparation tanks is ready to receive this material. In the preparation tanks 1, 6B and 7B, the coarse copper is dried or washed and dried before shipment to the smelter. These tanks are equipped with cocoa-matting filter bottoms of similar construction to those in the vats. Heavier lumber is used in these than in the vats.

Cement-copper settlers 8 to 16 are used for the settling of washed cement copper from the waste solutions. The total area of these is 1860 sq. ft. One of these, at least, is always out for digging. These sumps are decanted, and no pumping is required for this purpose.

When a double wash of the cement copper is employed, sumps 8 to 13 serve for the first washing and sumps 14 to 16 for the second washing. Except when employing the double washing scheme, the settling area for washed cement copper will be ample at maximum production.

The cuprous chloride settlers are served by one 10-ton Shaw crane and the cement-copper settlers by two 10-ton Shaw cranes. Bronze clamshell digging buckets are employed.

DISCUSSION

C. S. Witherell, New York, N. Y.—I believe we would like to hear more about the effect of molybdenum.

W. D. B. Motter, Jr., New York, N. Y.—The molybdenum is rather a hazardous proposition to discuss. We do not know a great deal about it ourselves. Apparently there is some effect caused by the molybdic oxide on the nitric acid which tends to break down and liberate NO. It seems to be encouraged in the presence of ferric iron also. There apparently is a series of reciprocating reactions in the tank houses as a result of this molybdenum. We are taking steps at the present time to evolve methods of handling the solution so as to eliminate molybdenum to a point where it can do no harm.

J. D. Sullivan, Tucson, Ariz.—Does the molybdenum in the solution affect both current efficiency and polarization?

W. D. B. Motter, Jr.—It affects current efficiency. Whether it affects polarization, I am not certain. Undoubtedly it has some effect. I think it is the NO. the liberation of the nitrous oxide.

Electrolytic Cadmium Plant of Anaconda Copper Mining Company at Great Falls, Mont.

By W. E. Mitchell,* Great Falls, Mont.

(New York Meeting, February, 1930)

Electrolytic production of cadmium at the Great Falls plant started in the first part of the year 1925. Prior to that time, an experimental unit had been in operation for a few months during the year 1922. At present the plant is the largest producer of cadmium, not only in the United States but in the world.

Electrolytic cadmium is characterized by its very low content of impurities. The main impurities, which are lead, copper, zinc and iron, do not aggregate more than 0.05 per cent., leaving a cadmium content better than 99.95 per cent.

The consumption of cadmium has increased rapidly since the war, with the growing use in the plating industry. Cadmium plating has come into prominence because of its connection with the automobile and radio industries. As a protective coating, a deposit of cadmium is superior to zinc.

Cadmium is also used extensively in alloys. Important among these is the alloy with copper used in telephone and trolley wire. In proportion of 0.5 to 1.2 per cent., cadmium adds materially to the strength and wearing qualities.

Practically all commercial zinc concentrates contain a small amount of cadmium. In the electrolytic zinc process, the cadmium that is dissolved in the regular leaching operation is removed from solution with metallic zinc, generally added in the form of zinc dust. This precipitate also contains the copper taken into solution and the excess of zinc dust, which is required in order to insure a complete removal of all impurities.

The precipitate is treated by a separate process to recover the copper, cadmium and excess zinc. In order to make the metals more soluble and to render insoluble such impurities as iron, arsenic and antimony, the precipitate is roasted. The roasting is carried out in gas-fired McDougall furnaces, with a roasting temperature of about 700° C.

The roasted residue is leached with dilute sulfuric acid, spent electrolyte from the zinc electrolyzing, containing 10 to 12 per cent. free H_2SO_4. The leaching is done in cylindrical tanks of 30 tons capacity, agitated

* Zinc Plant Superintendent, Anaconda Copper Mining Co.

mechanically. The leaches are finished neutral and decanted to Dorr thickeners. The thickened residue, still containing a considerable amount of copper, but with low cadmium and zinc content, is filtered and washed in a Moore filter, dewatered in an American filter, dried and shipped to a copper smelter. The overflow from the Dorr thickener contains copper, cadmium and zinc. Copper is first removed by a careful addition of zinc dust. The precipitation is carried out in tanks of 30 tons capacity, agitated mechanically. A high copper residue is produced, which is separated from the copper-free solution in a Dorr thickener, filtered in an American filter and shipped to a copper smelter.

The solution is treated with zinc dust, which precipitates cadmium in a finely divided state, contaminated by the excess zinc required to insure complete precipitation. The precipitation is carried out in tanks of the same construction and size as those used for copper. The precipitate, generally called cadmium sponge, is separated from the solution by filtration through a filter press.

Leaching of Cadmium Sponge

The cadmium sponge dissolves very slowly in dilute sulfuric acid. When oxidized, the solubility is greatly increased. The oxidation is carried out by stock piling the sponge for two or three weeks. The wet sponge oxidizes rapidly when piled in deep heaps, where a quick dissemination of the evolved heat from the oxidation cannot take place.

The partly oxidized sponge is leached with spent electrolyte from the electrolytic cadmium cells, in tanks of 30 tons capacity, agitated mechanically. The leaches are carried to the neutral point and allowed to settle in the leach tanks. The clear solution is filtered through a filter press. The filtered solution is pumped to a storage tank, from which it flows by gravity to the electrolytic cells.

The residue is allowed to remain in the leach tank until enough has accumulated to necessitate a clean-up. This residue and the mud from the filter press are returned to the plant treating the regular zinc-dust purification precipitate.

The advantage in using partly oxidized sponge is that it contains a sufficient amount of metallic zinc and cadmium to give a good purification. If the sponge is too high in copper, or has become completely oxidized, the solution may run too high in copper. In this case, a small addition of freshly precipitated sponge will be needed to precipitate rhe dissolved copper.

Electrolysis

Electrolytic deposition of cadmium is generally attended with a large production of trees and sprouts. Rotating cathodes are often used to overcome this tendency. The Great Falls plant, however, has always

used stationary cathodes, and has succeeded in establishing a condition that gives a satisfactory deposit with such an arrangement. The cells are of the same construction as for the electrolysis of zinc, and the same type of anodes and cathodes are used. Fig. 1 shows the sections of a cadmium electrolytic tank cell.

Each cell has 27 anodes and 26 cathodes. The spacing is 3.5 in. from center to center of cathodes. The cell voltage is approximately 2.6 volts and the current density is 4.25 amp. per square foot. Each cell has individual solution feed and the spent electrolyte discharges into a common collecting launder. The feed contains 100 to 120 g. per liter cadmium and 80 g. per liter zinc; the spent electrolyte about 70 to 80 g. per liter H_2SO_4. Glue is added to the cells to reduce the formation of beads. The amount used averages about 10 lb. per ton cathodes.

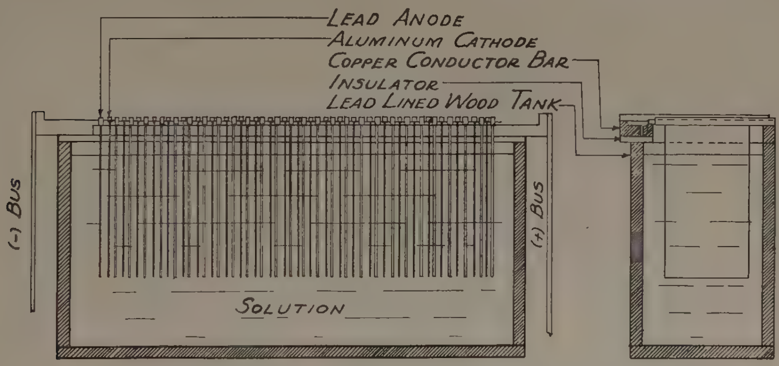


FIG. 1.—SECTION OF CADMIUM ELECTROLYTIC TANK CELL.

The tendency to form beads is also counteracted by carrying the cell temperature at about 35° C.

The cathodes are stripped every 24 hr. One plate is lifted out at a time and carried to the stripping rack. The beads are scraped off and treated separately. The sheets are washed with water, rolled into bundles and dried in a steam oven. The ampere efficiency based on the production of good sheets varies between 80 and 90 per cent. The percentage of metal produced as beads is generally about 5 per cent. The beads may be pressed together in a press and charged to the melting furnaces with the cathodes.

The spent electrolyte is returned to the tanks for leaching sponge. In order to keep down the zinc content, it is necessary to replace a certain amount of this electrolyte with fresh acid. Concentrated sulfuric acid may be used if pure contact acid is available; if it is not, spent zinc electrolyte, depleted to a low zinc content, may be used.

The cadmium in the acid that is withdrawn may be recovered by precipitation on metallic zinc. On account of the content of free acid,

the zinc consumption is high. A better way of disposal is to use the withdrawn acid to leach zinc-dust purification precipitate.

Melting the Cathodes

The dry cathodes are melted under a thin layer of caustic soda, to prevent excessive oxidation. The melting is carried out in cast iron pots of about 1000 lb. capacity, heated electrically to a temperature of 400° to 450° C. The pots are well hooded to prevent oxide fumes from entering the room. Fig. 2 shows a section of the melting pot.

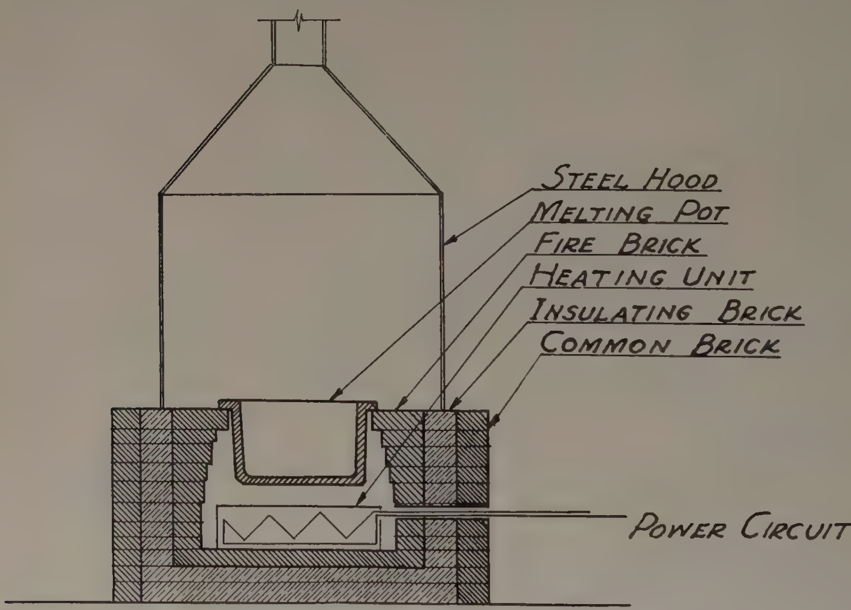


Fig. 2.—Section of cadmium electrical melting pot.

The caustic soda becomes thick and viscous from oxidized cadmium and is replaced occasionally. The consumption of caustic soda is about 40 lb. per ton of metal cast.

The caustic slag, containing cadmium oxide and a large amount of metallic beads and shots, is leached with water to dissolve the excess caustic soda. The solution is decanted off and the oxide and beads are treated with warm dilute sulfuric acid. The metal is taken into solution very slowly, because of its high purity. By the addition of manganese dioxide, however, the speed of the reaction may be greatly increased. The leach solution, after filtration, is fed to the electrolytic cells.

The cadmium is tapped from the pots and cast in bars weighing approximately 75 lb. These bars are remelted and the cadmium cast

into marketable shapes. At present, the cadmium is cast into pencils, slabs, anodes and balls. The pencils are ½ in. dia. by 10 in. long. Slabs are ¾ in. thick, 4 in. wide and 15¾ in. long. Anodes are ¾ in. thick, 2¾ in. wide and 17¼ in. long. The balls are 2 in. dia. These finished products are washed in dilute sulfuric acid, then in water, to remove the oxide film. They are then dried and packed into boxes for shipment.

DISCUSSION

H. R. Hanley, Rolla, Mo.—Has the author ever observed thallium in the cadmium? I know that thallium is a metal that frequently occurs in the analytical production of cadmium and must be removed.

F. G. Breyer, New York, N. Y.—I want to put in a word for cadmium. There seems to be a tendency to believe that cadmium will go on the downward track in the market, but there is an outlet for cadmium that perhaps many do not appreciate, and that is in the form of cadmium yellow. When I was associated with the New Jersey Zinc Co. during the war, the supply of straight cadmium sulfide was cut off from abroad and a number of people who were seriously concerned about making high-grade paints, particularly artists' colors that would last, made a strong plea to us to make some cadmium sulfide. We thought that if we could make the color, we might as well go one better and make it a little cheaper, so instead of making cadmium sulfide, we proceeded to make cadmium lithopone; we not only made a cheaper product but we made it just as strong, with only 35 per cent. cadmium sulfide, as the imported straight cadmium sulfide. The demand for metallic cadmium has caused the abandonment of the production of that material by at least one of the companies that was producing it. There is a definite need for that material in the paint trade. While the price has been high for ordinary paint purposes, it is surprising how much dollar-a-pound pigment is used in ordinary commercial painting. For example, many of the red gasoline pumps are painted with pigment that sells from $1.50 to $2 per pound. Cadmium lithopone can be made and sold for approximately 50 c. per pound, so there is no drawback to it.

C. P. Linville, Bound Brook, N. J.—I have been harping on this for several years in Mineral Industry. About 1900, the amount of cadmium made and used in the world was practically nil. The manufacture of cadmium has increased enormously since that time. New uses have been found for it, so much so that last year the price of cadmium was higher than it had been in years preceding, with a much increased amount of cadmium produced. It is a case where uses have been found for the metal practically as fast as the metal has been produced.

A Petrographic Study of Lead and Copper Furnace Slags

By Roy D. McLellan,* Maurer, N. J.

(New York Meeting, February, 1930)

The slags derived from the smelting of lead and copper ores are composed essentially of silicates. The problems arising from the smelting of these ores consequently involve the study of silicate fusions.

In the absence of specific knowledge concerning the compounds occurring in slags, it has been the practice of metallurgists to treat them in terms of their total chemical composition. Slags have been classified as monosilicates, bisilicates, sesquisilicates, and subsilicates according to the relative amounts of oxygen in the acid and basic radicles of the hypothetical salt produced.

While this has been a useful guide to metallurgists in computing the furnace charges, it falls far short of a basis for the scientific investigation of slags. In order to understand the nature of a slag as a whole it is necessary to learn the physical and chemical properties of the mineral compounds which go to make up that slag.

The efficiency with which an ore is smelted in a furnace depends primarily on the physical properties of the compounds formed during the operation. When the melting points and fields of stability of each of the mineral compounds in slags have been established, it will be possible to determine the properties of any given slag by examining the nature and amounts of its mineral constituents.

Earlier Investigations

In 1884 and succeeding years an extensive survey of lead and copper furnace slags was made in Europe by Vogt, and a large part of the present knowledge of the subject is due to his publications.[1] Vogt's interest in artificial fusions was directed primarily towards the explanation of problems in igneous and economic geology and his subsequent works have been confined wholly to that subject.

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* Research Department, American Smelting & Refining Co.

[1] J. H. L. Vogt: Studier over Slagger. Svenska Vet. Akad. Handl., 1884.

Beiträge zur Kentniss der Gesetze der Mineraldildung in Schmelzmassen und Ergussgesteinen. *Archiv. für Mathematick og Naturvidenskab.* Kristiania (1888–90) **13, 14.**

Die Silicatschmelzlösungen, I and II. *Kristiania Videnskabs-Selskap* (1903–04).

Many valuable papers dealing with particular phases of the lead and copper slag problems have appeared from time to time but a comprehensive study of the whole subject has never been made, and the literature on lead and copper furnace slags is still vague and in many cases contradictory.

While Vogt is to be credited with the first comprehensive survey of the slag problem, the establishment of the groundwork upon which the future study of slags must rest is due largely to the remarkable achievements of the Geophysical Laboratory of the Carnegie Institution of Washington. This investigation of the possible binary and ternary compounds, together with their fields of stability, melting points, and other physical data, in the ternary systems CaO-Al_2O_3-SiO_2, CaO-Al_2O_3-MgO, MgO-Al_2O_3-SiO_2, and CaO-MgO-SiO_2 will always stand as a monument and as a foundation for studies in slags.[2]

Numerous papers presented by the members of the Geophysical Laboratory, U. S. Bureau of Mines, and others have added to the knowledge of the properties of the various binary and ternary compounds. It is interesting to note that the experiments of the Geophysical Laboratory were made with the primary purpose of throwing more light on problems in igneous geology.

Unfortunately for the students of lead and copper slags, little is known concerning systems involving iron as one of the components. By the use of carefully controlled electric furnaces and containers of platinum for the fusions, a degree of accuracy was attained with the ternary systems mentioned that could not be hoped for in systems involving iron. At the temperatures necessary for their fusion, no known substance is able to resist corrosion or contamination by iron-bearing slags.

Another and much more serious problem in silicate fusions containing iron is the difficulty and necessity of maintaining a well-controlled oxidizing, reducing or neutral environment at high temperatures during the fusions.

It is safe to say that it will be a great many years before the systems involving iron are known to the extent that obtains with the other slag constituents.

Iron Furnace Slags

In the iron industry it is the purpose of the operators to keep the iron from entering the slag. These slags consist essentially of SiO_2, Al_2O_3,

[2] G. A. Rankin and F. E. Wright: The Ternary System CaO-Al_2O_3-SiO_2. *Amer. Jnl. Sci.* [4] (1915) **39**, 1–79.

G. A. Rankin and H. E. Merwin: The Ternary System CaO-Al_2O_3-MgO: *Jnl.* Amer. Chem. Soc. (1916) **38**, 568–588.

The Ternary System MgO-Al_2O_3-SiO_2: *Amer. Jnl. Sci.* (1918) **45**, 301–325.

J. B. Ferguson and H. E. Merwin: The Ternary System CaO-MgO-SiO_2. *Amer. Jnl. Sci.* (1919) **48**, 81–123.

CaO, and MgO. The investigation of the ternary systems involving the different combinations of these oxides, as first carried out by the Geophysical Laboratory, immediately placed the study of iron furnace slags on a working basis.

Students of iron furnace slags have recently extended the investigation to include the phases and their fields of stability in the quaternary system involving the oxides SiO_2, Al_2O_3, CaO and MgO.[3]

More and more is being learned about the conditions under which the different mineral compounds are formed or decomposed. The character of the slag as a whole is being constantly correlated with the percentages of its mineral constituents. The investigation of iron furnace slags with the help of the petrographic microscope has already become an exact quantitative science.

Lead and Copper Furnace Slags

The investigation of lead and copper furnace slags involves the study of the possible compounds resulting from fusions containing SiO_2, Al_2O_3, CaO, ZnO, sulfur and the oxides and ferrites of iron.

At first sight this might appear to be a hopeless task, and if the entire system involving these components were to be examined with the precision obtained in the examination of iron furnace slags it would indeed be discouraging. If the student of these slags should retire to his laboratory to synthesize and make a systematic investigation of the possible components involved in them, practical results might not be forthcoming during the present generation.

In many respects the slags from lead and copper furnaces present a close analogy with igneous rocks. They differ from iron slags in the fact that they are not "dry melts." Whereas the molten igneous rocks are charged with superheated steam, chlorine, and other mineralizers, the lead and copper slags are charged with nascent sulfur and the oxides of sulfur. This principally affects the iron contents of the slag. Iron-bearing compounds, which in a dry melt would be exceedingly refractory, are rendered easily fusible by the presence of nascent sulfur and its oxides.

For practical purposes the presence of mineralizers in lead and copper furnace slags tends to simplify rather than complicate the problem. Because of the strongly reducing action of sulfur, the iron entering the slag as silicate is largely in the ferrous condition. The percentage of ferric silicate that can exist in a workable slag is automatically limited by the temperature of the furnace. The formation temperatures of ferric

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[3] R. S. McCaffery, J. F. Oesterle and L. Shapiro: Composition of Iron Blast Furnace Slags. A. I. M. E. *Tech. Pub.* 19 (1927).

silicates are far above that reached in any lead-smelting or copper-smelting furnace.

The investigations that have been made on systems involving fusions containing SiO_2, Al_2O_3 and CaO can be used to form a basis for the study of lead and copper slags.

A superficial examination of the behavior of FeO in slag silicates under moderately reducing conditions would indicate a close analogy with that of MgO. The resulting compounds, however, are much less refractory.

When all of the available data on the components pertaining to lead and copper slags have been accumulated, it appears that the next step in the investigation is to determine by analytical methods the mineral constituents existing in commercial slag. Instead of attempting to cover the whole field with systematic synthetic fusions, let us restrict ourselves as much as possible to the portions of the system that include commercial slags.

After the various mineral compounds which occur in lead and copper slags have been determined by analytical methods, their conditions of formation and physical properties can be investigated by synthetic methods.

Analytical Methods

It is a relatively simple matter to examine a sample of slag and determine the composition of its mineral compounds. Thin sections prepared from slowly cooled slag samples can be examined with the petrographic microscope in the manner used in the study of rocks. Each of the mineral constituents can be separated by mechanical means and chemical analyses can be made on them. It is also possible to determine whether any mineral transformations have taken place during the cooling and crystallization of a slag. In other words, it is relatively easy to obtain complete analytical data on the chemical and mineral constitution of any given slag sample.

It is much more difficult and more important to determine the conditions that bring about the formation or decomposition of the various slag compounds. It is equally important to study the melting phenomena of each mineral and determine whether it forms solid solutions or reacts with certain constituents of the slag to produce other compounds.

Clean quick-running slags owe their properties to the formation of certain desirable mineral compounds. The ultimate goal of students of slags consequently is to produce the correct proportions of desirable minerals and avoid the formation of refractory compounds.

The Petrographic Microscope

It is fortunate that the students of slags have at their disposal the petrographic microscope, for without this instrument the study of slags

(as well as the study of rocks) would yield little information of value. Minerals which to the naked eye or under an ordinary microscope are indistinguishable can be readily differentiated from each other by means of the petrographic microscope.

There are many properties by which minerals may be differentiated. In instances where one or more properties fail to be diagnostic, certain other properties, or combinations of properties, will be distinctive. Among the outstanding diagnostic properties of crystals determined with the petrographic microscope and its accessories are the principal indices of refraction, the principal birefringences, the optic axial angle, extinction angles, optical character, elongation, color, pleochroism, and dispersion of the optic axes.

It must be emphasized that the most important property or constant of a given crystal is its chemical composition. In investigating a new and hitherto undescribed mineral, it is essential to find out exactly what it is composed of. Chemical analysis is the only method by which this information can be obtained.

Having determined by chemical analysis the composition of a certain mineral, it is the function of the petrographic microscope to investigate the physical properties that will serve to distinguish that mineral from all others. After these constants have been established, the mineral can in the future be determined by the examination of its properties with the petrographic microscope.

If petrographic work is to attain its maximum efficiency, it is essential that the elemental chemical composition of the specimen be known before the microscopic investigation begins. The chemical analysis tells us the amounts of the various elements present in a specimen; the petrographic investigation has for its purpose the determination of the compounds into which the elements have combined.

The use of X-rays in mineral analysis is yielding more and more information concerning the ultimate constitutions of crystals. The study of crystals by means of X-rays depends on the petrographic microscope in much the same manner that the latter depends on chemical analyses. For the most efficient application of X-rays to the study of minerals, it is important to begin with all of the chemical and petrographical information obtainable.

The success or failure of a metallurgical operation depends on the compounds formed during the process. The identification and the study of the properties of each compound is accomplished largely by the use of the petrographic microscope and its accessories.

Commercial Slags

Instead of attempting to study the entire system involving the components that occur in lead and copper slags, let us make a survey

of existing commercial slags and determine the nature of the compounds in them. Let us attempt to define the essence of the problem in each type of furnace operation and avoid the study of minor ingredients which appear to play no vital part.

To begin with, the furnace charge consists of an aggregate of minerals which have to be decomposed and converted into other mineral compounds in the furnace. Therefore it is necessary to study the melting phenomena of the minerals making up the furnace charge. Fortunately the number of ore and gangue minerals that occur in sufficient quantity to merit a detailed study is small. A large amount of information on the melting phenomena of many of the more important mineral groups is already available.

Copper Converter Slags

Copper converter slags will be considered first because of their relatively simple constitution and because their mineral ingredients have an important relation to the compounds in the more complicated slags.

Copper matte, consisting largely of copper and iron sulfides, is charged into the converter in the molten state. Quartz, or a siliceous rock, is added for the purpose of combining with the iron in the matte and causing it to form a slag. This is poured off at intervals and a new charge of quartz rock is added. A large quantity of air is blown into the molten matte by means of tuyeres, and the oxidation reactions supply the heat necessary to carry out the converting operation. When the iron has been removed from the matte by combination with the siliceous flux, the remaining "white metal," or Cu_2S, is "blown down" to blister copper by oxidation of the sulfur with a blast of air.

The slag produced in a converter consists almost entirely of iron and silica. It appears logical to assume, therefore, that a study of the system involving iron oxides and silica would completely define the converter slag problem.

The earlier literature dealing with iron silicates is discussed at length by Mellor.[4]

Four groups of modern workers on phase equilibrium in the system FeO-SiO_2 who stand out above the others are Greig, Herty and Fitterer, Whiteley and Hallimond, and Keil and Damman.[5]

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[4] J. W. Mellor: Inorganic and Theoretical Chemistry, **6**, 905.

[5] J. W. Greig: Immiscibility in Silicate Melts. *Amer. Jnl. Sci.* (1927) **13**, 133–154. On Liquid Immiscibility in the System FeO-Fe_2O_3-Al_2O_3-SiO_2. *Amer. Jnl. Sci.* (1927) **14**, 473–484.

C. H. Herty, Jr. and G. R. Fitterer: The System Ferrous Oxide-Silica. *Ind. & Eng. Chem.* (1929) **21**, 51–57.

J. H. Whiteley and A. F. Hallimond: Jnl. Iron and Steel Inst. (1919) **99**, 199.

O. von Keil and A. Damman: *Stahl und Eisen* (1925) **45**, 890

Fig. 1 shows the binary system FeO-SiO_2 as outlined by Herty and Fitterer. The construction of the more siliceous portion of the field is based on the work of Greig. This diagram may be considered to represent the most reliable existing data on the binary system FeO-SiO_2.

A single compound fayalite, $2FeO.SiO_2$, is shown in the diagram as forming eutectics with both FeO and SiO_2. The iron-rich eutectic contains 22 per cent. SiO_2 and melts at 1240° C., while the other eutectic contains 35 per cent. SiO_2 and melts at 1260° C. In applying this diagram to converter slags it should be advantageous to so regulate the slag composition that one of the eutectics would result.

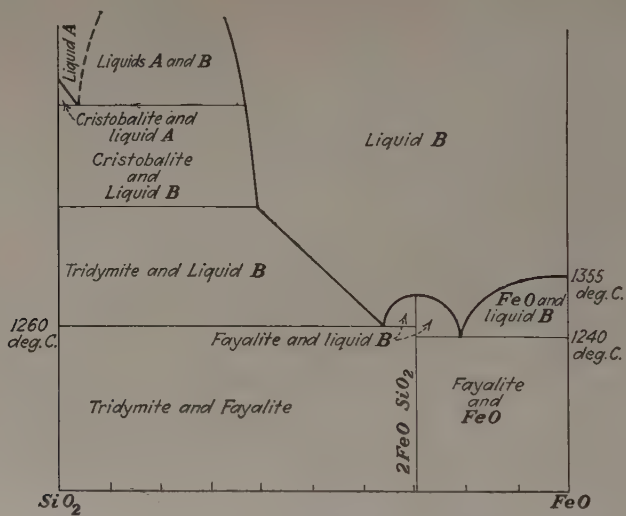


FIG. 1.—SYSTEM FeO-SiO_2 AS OUTLINED BY HERTY AND FITTERER.

The examination of many converter slags from different smelters has revealed the fact that they all consist essentially of magnetite, $FeO.Fe_2O_3$, and a ferrous silicate. The magnetite is never pure, but contains in solution small amounts of other members of the spinel mineral group. It always contains sulfur, presumably in the form of dissolved FeS. Silica, which probably occurs as dissolved ferrous silicate, is another consistent impurity in the magnetite.

The presence of both dissolved and mechanically intergrown FeS appears to be a universal occurrence in all ferrites from lead and copper furnace slags.

The ferrous silicate in converter slags always approaches the composition $4FeO.3SiO_2$ regardless of the proportions of the siliceous rock used as a flux for the iron. In all of the detailed investigations by the author on converter slags which contained insignificant amounts of CaO, Al_2O_3, or other impurities, the silica contents of the ferrous silicate did not vary

more than a few tenths per cent. from the theoretical composition $4FeO.3SiO_2$, which calls for 38.6 per cent. SiO_2. In each case the ferrous silicate particles were "hand-picked" and examined for impurities under the microscope. The hand-picked particles were ground exceedingly fine and the resulting product was subjected to repeated treatments with a powerful magnet to remove traces of finely divided magnetite.

Quenched samples of thoroughly smelted slag yielded a ferrous silicate identical in composition with that derived from ferrous silicate crystals produced from a slowly cooled crystalline slag.

Partly smelted converter slag samples contain some free silica but show no evidence of any mineral intermediate in composition between SiO_2 and $4FeO.3SiO_2$. Partly smelted slags, however, do show siliceous reaction rims surrounding the magnetite areas (Fig. 5). In such cases the reaction rim solidifies as a glass consisting largely of silica, in which semicolloidal magnetite is suspended.

In a recent paper by Wartman and Oldright[6] the principal reactions between magnetite and FeS in the temperature range of 1000° C. to 1300° C. are listed as follows:

$$FeS + 3Fe_3O_4 = SO_2 + 10FeO$$
$$FeS + Fe_3O_4 = S + 4FeO$$
$$FeS + 4Fe_3O_4 = SO_3 + 13FeO$$

The magnetite crystals in converter slags contain FeS, either mechanically intergrown or in solution. At the temperature of the furnace (1150° to 1200° C.) this FeS should react with magnetite to form FeO and an oxide of sulfur. The problem of the disposal of the liberated FeO now confronts us. Under reducing or neutral conditions the FeO would no doubt all combine to form ferrous silicate.

In the converter the reactions of the iron are controlled by the relative concentrations of silica, sulfur and oxygen. Sulfur is the reducing agent, and it must work in conjunction with the silica in order that a ferrous silicate may form. Ferric silicates require exceedingly high temperatures for their formation and fusion.

In other words, the formation of ferrous silicate in the converting operation depends on the efficiency of the reducing action of the sulfur at the instant the liberated iron is exposed to the action of silica. The iron must be allowed only sufficient oxygen to form FeO. The formation of magnetite results from the failure to maintain a sufficient degree of reduction at any given point of reaction.

The converting of copper matte into blister copper is essentially an oxidizing process. A large quantity of air is forced through the molten

[6] F. S. Wartman and G. L. Oldright: The Reaction between Magnetite and Ferrous Sulphide. *Rept. of Investigations* 2901, U. S. Bur. Mines (1928).

matte and slag. As the process continues, the concentration of sulfur (and particularly nascent sulfur) is constantly decreased. At the same time the concentration of the iron is also being decreased, for the slag is removed at intervals and a fresh charge of silica is added to a matte which is being constantly enriched in Cu_2S and depleted of FeS.

In the reaction centers in which the oxygen overcomes the reducing action of the sulfur, the FeS in the matte is oxidized to FeO and then further oxidized to magnetite, $FeO.Fe_2O_3$. At an early stage of the converting process, when the concentration of FeS in the matte is still great, the reactions listed by Wartman and Oldright work towards the decomposition of the magnetite already formed. In the oxidizing environment of the converter, however, it requires an immediate contact of the liberated FeO with silica to prevent the formation of magnetite.

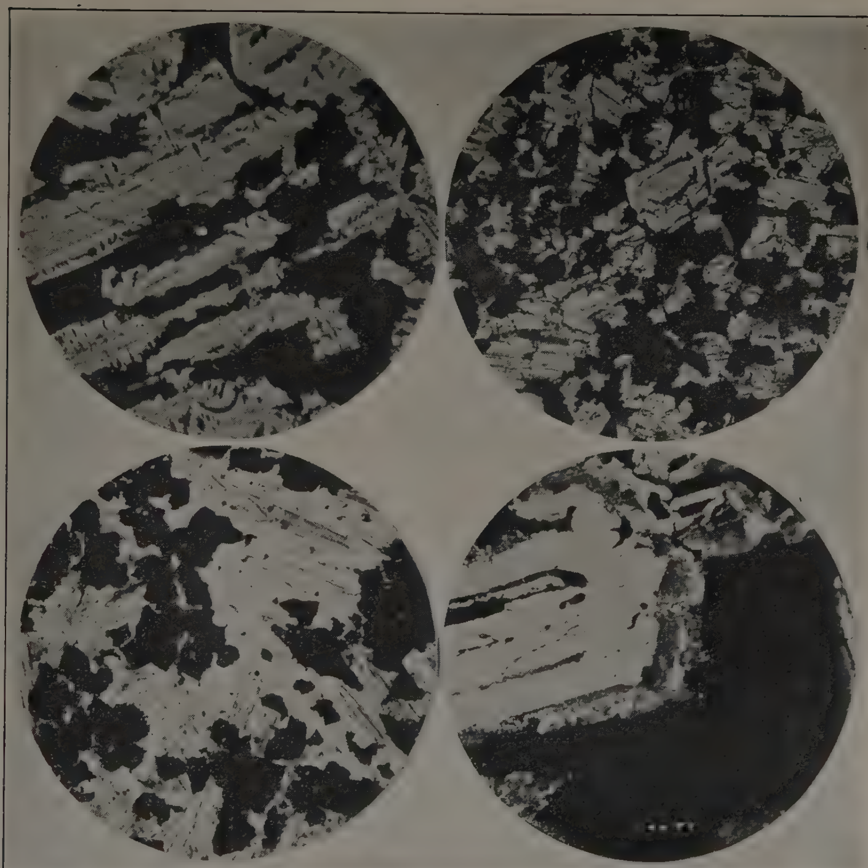
When magnetite is once formed in the converter, there is little hope for more than a minor amount of decomposition. Since the decomposition of magnetite depends primarily on the concentration of nascent sulfur in the presence of available silica, it becomes more and more stable as the converting operation progresses. The examination of thin sections of partly smelted converter slags taken at different stages in the process shows that the reaction rims surrounding the magnetite areas are much more noticeable in the early skims, and practically disappear at the finish. It is assumed that the semicolloidal magnetite occurring in the reaction rims is due to the oxidation of part of the FeO produced by the reaction between FeS and magnetite.

Nature of Ferrous Silicate in Slag

Experiments have shown that the ferrous silicate formed in the converter slag possesses a constant chemical composition of exactly, or almost exactly, 61.4 per cent. FeO and 38.6 per cent. SiO_2. Is this a definite compound, $4FeO.3SiO_2$, or is it a solid solution of two or more compounds?

The ferrous silicate in converter slags readily forms crystals belonging to the orthorhombic crystal system. The thin platelike crystals, enormously elongated in the plane at right angles to the acute bisectrix, are amber colored in thin section. The indices of refraction are somewhat lower than those of fayalite, $2FeO.SiO_2$. The texture and crystal habit changes considerably as the converting process goes on from start to finish (Figs. 2 to 4). Presumably this change is due to the gradually diminishing concentration of the oxides of sulfur. Skeleton forms resembling fayalite are common in converter slags but that is as far as the likeness goes.

When a large mass (10 or 15 tons) of converter slag is allowed to cool and crystallize without being disturbed, the pressure of entrapped SO_2,



2 3

4 5

Fig. 2.—Typical converter slag at beginning of converting process. × 50. Ordinary light.

SiO_2, 24.0 per cent.; Fe, 50.9; CaO, 0.9; Al_2O_3, 1.6; Sulfur, 2.6. Dark areas contain some unsmelted siliceous glass surrounding magnetite crystals. Light crystalline areas have the approximate composition $4FeO.3SiO_2$.

Except where otherwise stated, slag specimens illustrated were derived from 35-lb. samples which had been allowed to cool slowly.

Fig. 3.—Converter slag from intermediate skim. × 50. Ordinary light.

SiO_2, 22.0 per cent.; Fe, 51.0; CaO, 0.9; Al_2O_3, 1.8; sulfur, 0.7. Unsmelted siliceous glass had been largely eliminated from this slag. Specimen was taken from same furnace charge as Fig. 2.

Fig. 4.—Converter slag from last skim before "going on finish." × 50. Ordinary light.

SiO_2, 21.0 per cent.; Fe, 42.5; CaO, 0.8; Al_2O_3, 2.0; sulfur, 0.21. Specimen was taken from same furnace charge as Figs. 2 and 3. Shows typical texture of ferrous silicate at this stage of process. Dark areas are crystalline magnetite. Light areas are ferrous silicate with approximate composition $4FeO.3SiO_2$.

Fig. 5.—Converter slag from 10-ton sample which had been allowed to cool slowly. × 50. Ordinary light.

Finely divided magnetite occurs as suspension in highly siliceous glass which surrounds the large magnetite (black) areas. Ferrous silicate crystals are sharply divided from glassy areas.

and perhaps other gases, is apparently great during the later stages of the crystallization. The crystals lining the internal cavity in such a cooled mass of slag have the external form of melilite crystals. Optically their properties are identical (at least when cold) with the normal slag silicate crystals.

The examination of lead blast-furnace slags indicates that a close relationship exists between the converter slag silicate and the pyroxene mineral group on one hand and the melilite group on the other.

The ferrous silicate in converter slags melts with a fairly abrupt softening range at approximately 1125° C. It forms a thinly fluid melt at the lowest temperature of the converter, or about 1150° C. If this silicate is a simple binary it should, according to Herty and Fitterer, melt at about 1300° C. This temperature is never reached in the converter. It appears, therefore, that the ferrous silicate in converter slags is not a simple binary composed of silica and FeO. Sulfur, as explained below, appears to be the third essential component, though its amount is relatively very small.

Due primarily to the presence of sulfur and its oxides, the magnetite in converter slags occurs in the molten state at a temperature lower than 1200° C. The fact that magnetite crystals after separation from the slag will not melt below 1350° C. to 1400° C., would indicate that volatile constituents play an important part in maintaining its fluidity at lower temperatures.

By the same token, the fact that separated ferrous silicate crystals can be remelted at a temperature of approximately 1125° C. would indicate that it contains a stable third component in addition to ferrous oxide and silica. Sulfur, which persistently remains to the extent of a few tenths per cent., appears to answer the description of this third component.

While the determination of the ultimate constitution of the converter slag ferrous silicate must depend on X-ray studies, the observations with the petrographic microscope favor the idea that it is fayalite, $2FeO.SiO_2$, saturated with dissolved silica and FeS.

If the converter is allowed to blow after all of the silica has been converted into ferrous silicate of the composition mentioned, the remaining FeS in the matte can only oxidize to form magnetite. This process continues until the magnetite crystals become so abundant that they interlock with one another (Fig. 6).

At this point the walls of the furnace begin to take on a coating of magnetite in which the interstices between the crystals are filled with normal converter slag ferrous silicate. The nature of the minerals in the furnace lining is the same as that of the constituents of a normal fluid slag. Only the proportions of these ingredients have been changed.

Copper Blast-furnace Slags

The normal or usual type of copper blast-furnace slag consists largely of the converter slag ferrous silicate modified by solid solutions involving lime, alumina and other ingredients. The additional components are not usually present in sufficient amounts, and the environmental conditions have not been sufficiently changed, to destroy the crystal habit of the silicate.

Copper blast-furnace slags possess a crystalline texture when slowly cooled that is almost universally characteristic of such slags (Fig. 7). Thin sections of these slags show a reticulated texture resembling that of certain basaltic lavas. When considered in terms of three dimensions, however, the texture is quite different from that of ophitic basalts. It consists of thin platelike silicate crystals enormously elongated in a single plane but without any systematic orientation of the plates with respect to one another. The interstices between the platelike crystals are filled with semicrystalline silicates, usually having a relatively high content of alumina. Magnetite, and other dissolved ferrites, are particularly characteristic of these interstitial regions (Fig. 8).

For reasons not yet understood, the silicate solutions in copper blast-furnace slags occasionally break up to form the olivine mineral fayalite, $2FeO.SiO_2$; the pyroxene mineral hedenbergite, $CaO.FeO.2SiO_2$; and the melilite mineral gehlenite, $2CaO.Al_2O_3.SiO_2$ (Fig. 9).

In this case the fayalite crystals possess all of the optical properties of the pure mineral and their chemical composition coincides with the formula $2FeO.SiO_2$. This type of slag may continue for several days and then suddenly revert to the normal type without any apparent change in the total chemical composition of either the furnace charge or the slag.

The ingredients of a blast-furnace charge are so situated that there is an ideal opportunity for the more easily fusible compounds and eutectics to melt and flow away from the remainder of the charge. Due to the viscosity of silicate melts, a large part of the residual unmelted or partly melted portions of the furnace charge is carried off in suspension in the slag.

The nature of the residuals varies with the chemical and mineral composition of the furnace charge. It also is a function of the maximum temperature attained in the furnace and the length of time it was subjected to that temperature.

Copper Reverberatory-furnace Slags

While the slags from copper reverberatory furnaces may have the same chemical composition as those from copper blast furnaces, the texture and apparently the nature of the solid solutions is distinct.

6 7 8 9 10 11

FIGS. 6 TO 11.—LEGENDS ON OPPOSITE PAGE.

In nearly all cases, however, the converter slag silicate forms the basis for the solution of other components of the charge. In fact, the great flexibility of the reverberatory furnace in its capacity to smelt charges with considerable variations in composition depends on the ability of the ferrous silicate to dissolve large amounts of many other compounds without seriously changing the melting properties.

Equilibrium, a condition that is usually far from being reached in lead or copper slags, is more nearly approached in the reverberatory than in blast furnaces. Eutectics and easily fusing compounds are unable to melt and flow away from the remainder of the charge to the extent seen in blast-furnace slags. Even in the reverberatory furnace, however, the ingredients of the charge that dissolve less readily in the more easily fusing portions of the slag tend to concentrate in the furnace as residuals and gradually slow up the speed of smelting.

A given furnace charge may smelt with great rapidity for several days. At the end of a week the same charge may smelt much less readily, though no change in the composition of the charge has occurred. The reason for this behavior is the gradual building up of materials which do not dissolve readily in the more easily fusing portions of the slag.

By changing the composition of the charge it may be possible automatically to increase the capacity of the fluid portions to dissolve the residuals and remove them from the furnace as slag. If no change is made in the composition of the furnace charge, the residuals may be

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FIG. 6.—CONVERTER SLAG BLOWN BEYOND STAGE AT WHICH FREE SILICA HAD BEEN USED UP. × 50. ORDINARY LIGHT.

SiO_2, 15.0 per cent.; Fe, 58.8; CaO, 0.2; Al_2O_3, 0.6; sulfur, 3.1. Black areas are impure magnetite with inclusions of matte. Light areas are crystalline ferrous silicate having approximate composition $4FeO.3SiO_2$. Specimen has general appearance and composition of basic converter lining.

FIG. 7.—COPPER BLAST-FURNACE SLAG. × 50. ORDINARY LIGHT.

SiO_2, 38.7 per cent.; Fe, 28.5; CaO, 10.6; MgO, 1.6; Al_2O_3, 6.8; sulfur, 0.3. Texture of this specimen is typical of these slags. Slag consists largely of modified ferrous silicate (white) occurring as drusy plates whose edges are seen in the figure. Interstices (black) between platelike crystals are either glassy or finely crystalline, depending on viscosity of slag.

FIG. 8.—COPPER BLAST-FURNACE SLAG. × 75. ORDINARY LIGHT.

Dendrites of impure magnetite (black) in finely crystalline interstitial areas between plates of ferrous silicate. Rodlike white areas are edges of platy crystals.

FIG. 9.—COPPER BLAST-FURNACE SLAG. × 50. ORDINARY LIGHT.

SiO_2, 38.8 per cent.; Fe, 28.5; CaO, 10.8; MgO, 1.8; Al_2O_3, 7.1; sulfur, 0.2. Slag consists of fayalite, $2FeO.SiO_2$, and gehlenite, $2CaO.Al_2O_3.SiO_2$ (both white) in groundmass of hedenbergite, $CaO.FeO.2SiO_2$ (mottled). Notice that chemical composition is almost identical with that of Fig. 7.

FIG. 10.—COPPER REVERBERATORY-FURNACE SLAG. × 50. ORDINARY LIGHT.

SiO_2, 40.8 per cent.; Fe, 29.5; CaO, 5.5; MgO, 1.1; Al_2O_3, 6.4; sulfur, 0.2. Ferrous silicate modified by small amounts of CaO, MgO, and Al_2O_3 occurs as bundles of more or less parallel platelike crystals whose edges are seen in figure (white). Interstitial material is glass and impure magnetite (black).

FIG. 11.—COPPER REVERBERATORY-FURNACE SLAG. × 50. ORDINARY LIGHT.

SiO_2, 38.4 per cent.; Fe, 30.1; CaO, 6.6; MgO, 1.4; Al_2O_3, 5.5; sulfur, 0.2. Similar to Fig. 10, except that interstitial material is finely crystalline instead of glassy. This slag was less viscous than the other, as shown by more coarsely crystalline texture of platelike crystals.

caused to dissolve by allowing the furnace to run for several hours without charging or tapping.

The presence of alumina or alkalies in the furnace charge appears to aid the solution of the other ingredients in the converter slag silicate. This may or may not increase the melting point of the fluid slag, but at any rate it will tend to be more homogeneous. The tendency to form residuals when moderate amounts of alumina or alkalies are present is considerably reduced. At the same time the viscosity of the slag may be increased.

It is interesting to note that these observations are somewhat analogous to the effect of alumina and alkalies on liquid immiscibility as studied by Greig.[7]

The modified converter slag silicate occurring in reverberatory-furnace slags crystallizes in subparallel groups of thin platelike crystals (Figs. 10 and 11). These plates are always elongated in the plane at right angles to the acute bisectrix. They belong to the orthorhombic crystal system. The interstices between the plates are filled with ferrites and semicrystalline silicates which failed to dissolve in, or liquated from, the molten converter slag silicate solutions.

A number of different minerals have been identified as constituents of these poorly crystallized areas. Among these are hedenbergite, $CaO.FeO.2SiO_2$, melilites of the gehlenite-akermanite type with FeO replacing MgO, mullite, $3Al_2O_3.2SiO_2$, anorthite, $CaO.Al_2O_3.2SiO_2$, and members of the olivine mineral group. The ferrites are always composed largely of magnetite.

Lead Blast-furnace Slags

Lead blast-furnace slags usually contain a greater variety of free mineral compounds than any of the copper furnace slags. Occasionally the silicate portion of the slag consists of a single mineral, such as an olivine, a pyroxene, or a melilite mineral, but usually several of these mineral groups are represented in the same slag.

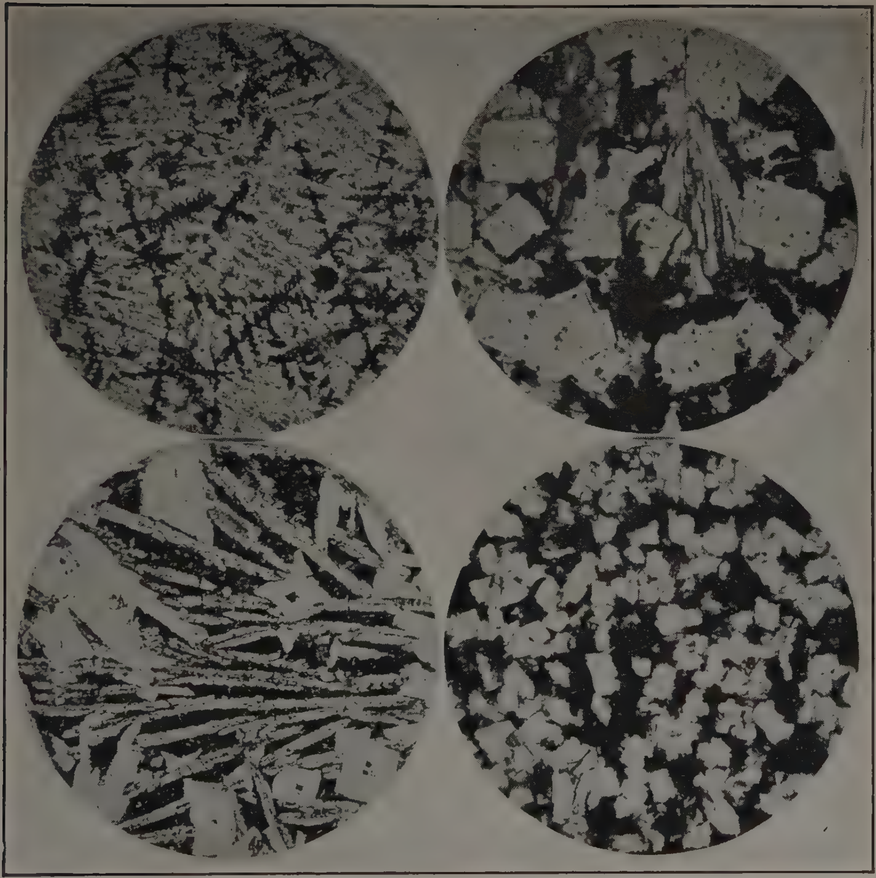
Presumably the olivine mineral may consist of the molecules indicated below, dissolved in one another.

Olivine Molecules	Melilite Compounds
$2FeO.SiO_2$	$2CaO.Al_2O_3.SiO_2$
$2CaO.SiO_2$	$2CaO.FeO.2SiO_2$
$2MgO.SiO_2$	$2CaO.MgO.2SiO_2$
$2ZnO.SiO_2$	$2CaO.ZnO.2SiO_2$
$2MnO.SiO_2$	$2CaO.MnO.2SiO_2$

The melilite mineral appears to be composed essentially of the compounds indicated, dissolved in one another. The pyroxenes seen in lead blast-

[7] J. W. Greig: Immiscibility in Silicate Melts. *Amer. Jnl. Sci.* (1927) **13,** 133.

12 13



14 15

FIG. 12.—LEAD BLAST-FURNACE SLAG. × 50. ORDINARY LIGHT.

SiO_2, 23.2 per cent.; Fe, 30.2; MnO, 1.4; CaO, 12.4; MgO, 1.2; Al_2O_3, 3.6; Zn, 12.8; sulfur, 3.4. Silicate portion of slag occurs entirely in form of pyroxene crystals (white). Ferrites (black) occur chiefly in form of crystalline dendrites. Some of gray mottled areas are composed of ZnS which, in this specimen, occurs chiefly in form of dendrites.

FIG. 13.—LEAD BLAST-FURNACE SLAG. × 50. ORDINARY LIGHT.

SiO_2, 24.0 per cent.; Fe, 28.5; MnO, 1.4; CaO, 14.0; MgO, 1.3; Al_2O_3, 3.5; Zn, 12.7; sulfur, 3.2. Specimen was derived from a 5600-lb. sample which cooled slowly in a slag car. Rectangular white areas are composed of iron-bearing melilite. Prismatic white areas and groundmass are pyroxene. Dark areas are chiefly ferrite, but contain scattered crystals and dendrites of ZnS.

FIG. 14.—LEAD BLAST-FURNACE SLAG. × 50. ORDINARY LIGHT.

SiO_2, 35.0 per cent.; Fe, 23.9; MnO, 0.7; CaO, 19.0; MgO, 1.1; Al_2O_3, 3.0; Zn, 6.2. White areas are iron-bearing melilite. Mottled rodlike areas are platy crystals of modified ferrous silicate. Dark interstitial material between platy crystals is composed of strongly pleochroic hedenbergite. Ferrites and ZnS are scarce in this specimen.

FIG. 15.—LEAD BLAST-FURNACE SLAG. × 75. ORDINARY LIGHT.

SiO_2, 32.9 per cent.; Fe, 26.6; MnO, 1.1; CaO, 16.9; MgO, 1.9; Al_2O_3, 2.9; Zn, 6.7. Silicate portion of this specimen (white) occurs almost entirely as an olivine mineral. Small amount of pyroxene mineral augite occurs in groundmass (black) together with ferrites and small quantities of ZnS.

furnace slags belong to the hedenbergite-augite division of the pyroxene mineral group.

A silicate mineral occurring in platelike crystals, which closely resemble those in copper converter slags, forms an important constituent of lead blast-furnace slags (Fig. 14). These crystals are particularly characteristic of slags from furnaces that produce matte.

Lead and copper slags are never free from minerals of the spinel group. Magnetite is the most abundant representative of the group and usually holds the other spinel molecules dissolved in it. The extended formula for magnetite is $FeO.Fe_2O_3$. When the FeO is replaced by ZnO, CuO, MgO, etc. a series of ferrites are obtained, which apparently are soluble in one another in all proportions. At the same time, the Fe_2O_3 may be partly or wholly replaced by Al_2O_3 or Cr_2O_3 to form additional spinel molecules. $ZnO.Al_2O_3$ is occasionally found in lead blast-furnace slags in free crystals. This is the only observed exception to the rule that the spinel molecules in these slags occur in solution in magnetite.

Were it not for the beneficial action of sulfur, the formation of minerals of the spinel group would present a difficult if not insurmountable problem in lead blast-furnace operation. If iron were not the preponderating constituent of the spinel minerals in such slags, the sulfur would be unable to produce such beneficial results in lowering the melting point below the temperature of the furnace. The presence of dissolved silica and sulfur (presumably in the form of FeS and ferrous silicate) appears to be universally characteristic of the ferrites in lead and copper furnace slags.

ZnS, and occasionally free CaS, are present in the typical lead blast-furnace slag.

Synthetic Fusions

The ore and gangue minerals in the furnace charge must be decomposed and the elements recombined in such a manner that the most desirable combination of mineral compounds will result. In order to study the behavior of the various ore and gangue minerals when melted in a furnace, it is necessary to make systematic synthetic fusions containing them. Fortunately the number of these minerals that occur in appreciable amounts is quite small.

Within the range of composition covered by commercial slags, and to some extent outside of it, all possible data on phase equilibrium must be obtained. The study of the binary systems involving all possible combinations of the phases that permit direct synthesis in the pure state can greatly simplify and direct the course of subsequent fusion experiments.

The conditions that produce a pyroxene mineral at one time and a melilite or an olivine at another, from chemical elements which apparently

are identical in quantity and kind, will require a detailed investigation by synthetic methods.

Slags from lead or copper furnaces seldom closely approach equilibrium with all parts of the furnace charge. The more easily fusible compounds and eutectics melt and often run away from the remainder of the charge. If given a sufficient length of time in contact with the charge as a whole, the easily fusible portions would dissolve or react with the remainder until eventually a homogeneous fluid slag would result. The time factor is an important one in all smelting operations.

Summary

The ease with which a given furnace charge will smelt depends on the physical properties of the compounds produced in the furnace. The nature of these compounds can be learned from the study of the slags. The ultimate goal for studies in slags is to bring about the formation of desirable compounds and prevent or reduce to the minimum the formation of undesirable ones.

The problems of lead and copper furnace slag are complicated by the difficulty in studying, under controlled conditions, silicate melts containing iron and sulfur as essential ingredients.

This paper has resulted from the study of thin sections, from a large number of commercial lead and copper furnace slags, with the help of the petrographic microscope. It is an analytical survey of these slag problems designed to simplify the course to be followed in future slag studies by synthetic fusions.

DISCUSSION

C. R. Hayward, Cambridge, Mass. (written discussion).—This paper is of considerable interest as a contribution to a subject regarding which too little is known. The work of the Geophysical Laboratory on silicates of alumina, lime and magnesia referred to by the author was a notable contribution to our knowledge of slags, and it is hoped that work of a similar nature may be carried out on mixtures containing ferrous oxide.

The author refers to the reactions between FeS and Fe_3O_4 as reported by Wartman and Oldright. I made a similar study in 1922, using pure artificial FeS in excess and natural magnetite crystals, also a number of tests using an excess of copper reverberatory matte and the magnetic slag formed by blowing matte without flux. The fusions were made in a platinum crucible. The reaction was violent between 1160° and 1200° C. with considerable ebullition and evolution of SO_2 gas. This confirms the statement referred to. It is undoubtedly true that for the most satisfactory slag-forming condition some silica must be available to react with the FeO liberated. I am not sure that I agree with the statement that sulfur is the reducing agent. Would it not be more accurate to say that FeS was the reducing agent, because the iron as well as the sulfur removes oxygen from the magnetite?

The author says (p. 252) that in copper converters if magnetite is once formed there is little chance of its being reduced. In view of the reactions referred to above I am

inclined to the belief that there may be considerable reduction. Reoxidation will take place, of course, if silica is not available to unite with the ferrous oxide.

The determination of the mineral constituents in the slags is of interest but the final mineralogical composition apparently depends on factors other than composition and the results seem therefore to be of little direct practical value. It might be helpful if the order of crystallization of these mineralogical constituents were determined. It is also possible that the proper approach to the greatly needed fusion tests would be to start with mixtures of some of these silicates. I think we have a start in this work of something which may go on perhaps to further petrographical study and finally tie up with some fusion tests, which of course must follow these experiments if the man who is actually operating the furnace is to be benefited.

C. P. Linville, Bound Brook, N. J.—These papers are interesting from the standpoint of slags that have frozen. The metallurgist is usually concerned with slags that are molten. I think that the physical chemist as a rule will say that the chemical compounds present in a frozen slag are perhaps a result of what was present in the liquid slag, but most of these compounds are formed at the time of freezing and they give practically no inkling as to what was the composition of the liquid slag itself. Metallurgists are interested in the properties of liquid slags and to the extent that petrographic work will give an inkling of what might have been the condition in the liquid stage, this paper is valuable. Melting points and freezing points may be very different. Once a slag has been formed cold with large crystals of high-melting-point material, it may take a long time for it to remelt and it may be different from the conditions that happen when the materials of the furnace charge themselves are melted before having gone through a freezing point.

C. S. Witherell, New York, N. Y.—Following up what Mr. Linville has just said, will the author kindly elaborate upon the effect he mentions in his paper, of sulfur and oxides of sulfur in lowering the melting point of the slags? I think that has a bearing on the subject.

R. D. McLellan.—The point was raised that FeS rather than sulfur was the reducing agent in slags. My intention was just to convey the idea that it was some sulfur compound. I believe that the oxides of sulfur, particularly the nascent oxides, are important in all of these slags. If you pick out the ferrites from slags and try to fuse them, you cannot do it at anywhere near the temperature at which the thin sections of slags show the ferrites to have been in the molten condition. Something in the way of a mineralizer apparently has been present in the molten slag. This is not just the opinion derived from the examination of one or two thin sections; it is derived from the observation of thousands of thin sections, taken where the furnace conditions were studied in connection with them.

C. R. Hayward.—Will you elaborate a little further on how the sulfur acts in lowering the melting point? Does it take a direct part?

R. D. McLellan.—I would say that the sulfur compounds are directly in solution, just as superheated steam, for instance, will dissolve the different silicates provided you can apply the superheated steam.

C. S. Witherell.—SO_2 would really be a constituent.

R. D. McLellan.—Yes, it is a constituent that is always lacking in the investigation with thin sections of the solid slag.

C. S. Witherell.—Apparently that will lower the temperature of the melting point some 150° C.

R. D. McLellan.—Yes.

C. P. Linville.—What is the point of dissolved gases in lowering the freezing point of water? For instance, is it not true that a certain amount of air, or carbon dioxide, dissolved in water will lower its freezing point to a certain extent? The energy involved in driving gases out of a solution at the freezing point would seem to me to allow a lowering of that freezing point. I merely bring up the question whether there are not other analogous things in solution that have gases dissolved which are liberated at the freezing point.

C. R. Hayward.—Is there a physical chemist here to elaborate on that?

C. S. Witherell.—Does not the author really mean something more than that, not merely occluded gas dissolved in the slag as water dissolves a gas but really a closer combination between the sulfur compounds that are gaseous in their free state and the other slag constituents than usually understood by dissolved gases? A lowering of the freezing point 150° C. or more would indicate such.

R. D. McLellan.—I do not know enough about it to finally answer the question, but the observed fact is there that the melting point is greatly reduced. However, the analogy is in igneous rocks. Take silica, for instance, which fuses at 1713° C. and yet it is true that a considerable proportion of the quartz in nature solidified as low as 573° C., due to the presence of mineralizers. This can be duplicated in the laboratory.

B. M. O'Harra, Maurer, N. J.—It might be interesting if Dr. McLellan would say a little about the manner in which zinc occurs in blast-furnace slags.

R. D. McLellan.—In the different blast-furnace lantern slides that were shown, we found that some of the minerals crystallized out as pyroxenes, some as melilites, some as olivine minerals. Although we may not ultimately be able to follow the slag clear back into the molten stage, we can carry the study as far as the latent heat of fusion at least, and if there is anything peculiar or abnormal about the energy effects shown by the latent heat fusion of any mineral, this study can be carried still farther. We should carry this investigation a little farther than has been done so far.

The occurrence of zinc in blast-furnace slags depends to a great extent on the nature of the silicate minerals. If a melilite mineral is formed, it is reasonable to believe that the proportion of the zinc occurring as silicate exists in the melilite as dissolved zinc akermanite; that is, $2CaO.ZnO.2SiO_2$. Zinc occurs always as zinc ferrite dissolved in magnetite. Magnetite in all of these slags, whether copper or lead, is the chief constituent of the ferrite. The ferrites will also contain dissolved alumina. They also contain sulfur. The last trace of sulfur cannot be eliminated by fusing. Usually there is zinc in the form of free zinc sulfide. If any part or all of the silicates in the slag crystallize out in the form of an olivine mineral, the same conditions exist as before, excepting that the silicate portion contains zinc in the form of dissolved willemite, $2ZnO.SiO_2$. Concerning the nature of the zinc dissolved in pyroxene, I suppose it is the unstable molecule $ZnO.SiO_2$ or perhaps $CaO.ZnO.2SiO_2$. There is no stable molecule having the composition $ZnO.SiO_2$. This is just a hypothetical way of explaining the occurrence of zinc dissolved in the pyroxene crystals. At any rate, when you separate the silicate crystals from a blast-furnace slag you will always find a considerable proportion of zinc occurring as silicate.

G. R. Fitterer, Pittsburgh, Pa. (written discussion).—The author's reference to and discussion of the system $FeO-SiO_2$ is particularly interesting in that the converter slags always approach a constant composition represented by the formula $4FeO.SiO_2$. Unquestionably, this is the so-called liquid B saturated with silica at the temperature involved.

A very similar situation occurs in acid open-hearth steel furnaces, in which a constant composition (55 to 60 per cent. SiO_2 and 15 to 20 per cent. each of FeO and MnO) is always approached. If a horizontal line is drawn across Fig. 1, at 1600° C., its intersection with the liquidus occurs at approximately 60 per cent. silica. This phenomenon represents a heterogeneous equilibrium between the oxide content of the liquid metal, the liquid slag phase and the solid silica in the furnace lining. If iron oxide is added to the slag, silica will be taken into the slag (by fluxing with the furnace lining) at the same time that the metal is being oxidized. In consequence, the above "constant" slag composition is again obtained.

Both of these silicate equilibria are controlled somewhat by the viscosity of the slags and any ambitious research program should include a study of that. For example, in the case of the copper converter slag the temperature-viscosity relationships of the base slag ($4FeO.SiO_2$) could be determined, after small additions of other materials such as FeS have been made.

The compositions of lead blast-furnace slags have recently been studied by G. L. Oldright and Virgil Miller.[8] Likewise, the composition of copper converter slags has been studied by F. S. Wartman and W. T. Boyer.[9] These investigations have been mentioned because they are pertinent to the subject matter discussed here and at the same time do not appear in any society publications. Incidentally, Carl Küttner[10] studied the system CuO-SiO_2 and found a striking similarity between it and the system FeO-SiO_2.

R. D. McLellan (written discussion).—Evidence from recent fusion experiments made in connection with converter-slag studies tends to show that we are dealing with a much more complicated system than at first appeared to be the case.

In a graphite crucible it is easily possible to synthesize $4FeO.3SiO_2$ from commercial FeO and SiO_2. In such an environment no magnetite is formed, and if the temperature is not carried too high or the melting period prolonged, there will be no reduction of iron to the metallic state. This simple binary mixture requires approximately 1300° C. for its fusion, a temperature that is in harmony with the results of Herty and Fitterer.[11] This melt is too viscous to permit the formation of good crystals even when cooled very slowly.

If a small amount (say 0.5 per cent.) of FeS is added to a melt of which the composition is represented by the ratio $4FeO.3SiO_2$ there is a remarkable increase in fluidity. The melting point is lowered more than 100° and even on moderately rapid cooling the crystallization is complete.

The examination of large samples of converter slag (15 tons), which had been allowed to cool and crystallize slowly, has recently revealed an unmixing of the solution to form a mechanical intergrowth consisting largely of $2FeO.SiO_2$, FeS, and silica in addition to the ever-present ferrites.

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[8] G. L. Oldright and V. Miller: U. S. Bur. Mines *Repts. of Investigations* 2966 (1929) and 2954 (1929).

[9] F. S. Wartman and W. T. Boyer: U. S. Bur. Mines *Repts. of Investigations* 2985 (1930).

[10] Unpublished Diplom. Arbeit, Breslau, 1927.

[11] C. H. Herty, Jr. and G. R. Fitterer: The System Ferrous Oxide-silica. *Ind. & Eng. Chem.* (1929) **21**, 53, Fig. 2.

Lead Refining at the Bunker Hill Smelter of the Bunker Hill and Sullivan Mining and Concentrating Co.

By Alfred F. Beasley,* Kellogg, Idaho

(New York Meeting, February, 1930)

Lead-refining practice at the Bunker Hill differs to some extent from that of other United States refineries using the Parkes process, in that the Bunker Hill has reverted to a custom used years ago of making two kinds of skims, or crusts, in the desilverizing kettles. Also, this was the first refinery to adopt the liquation process for silver skims as developed in Australia. Both of these ideas were brought to the attention of the operator by H. S. J. Sommerset, in the summer of 1924, while he was visiting metallurgical plants in the United States, when he was general superintendent of the Broken Hill Associated Smelters Pty., Ltd., of Port Pirie, South Australia.

Since Mr. Sommerset's visit, United States patents on the liquation process have been issued to George Kenneth Williams, of Port Pirie, South Australia.

At the Bunker Hill, the lead bullion is conveyed from the blast furnaces to the refinery in brick-lined cast-steel pots, each having a capacity of 5 tons, by a 20-ton electric crane, to which is suspended a Fairbanks-Morse suspension-type scale on which the gross, tare and net weights are obtained.

The bullion is poured from the small bullion pots into either one of two kettles of 105 tons capacity, and when a kettle is entirely filled the heavy dross is removed by skimming it into a 5-ton pot suspended by the crane. The crane operator becomes proficient in the manipulation and a kettle is skimmed in a very short time. After the kettle is skimmed clean the bullion is pumped to a kettle of the same size alongside, where the bullion is cooled until it has almost frozen. By using two kettles for the drossing operation, one to accumulate the bullion where the heavy dross is removed, and then pumping to a clean kettle comparatively free of hangings, etc., it is possible to reduce the copper content much lower than if only one kettle is used in the operation. The average yearly analysis of bullion is given in Table 1.

During July, August, September, and October, when a considerable quantity of high-grade silver ore from the Yukon Territory is smelted,

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* Superintendent, Bunker Hill Smelter, Bunker Hill & Sullivan Mining & Concentrating Co.

the analysis of the bullion is as follows: Au, 0.53 oz.; Ag, 190.0 oz.; Cu, 1.10 per cent.; Sb, 1.25 per cent.; As, 0.15 per cent.

After the copper has been removed from the bullion in the drossing operation, the bullion is heated to approximately 900° F. and pumped to any one of three softening furnaces through a trough lined with a mixture of two parts of cement and five parts of lime rock, crushed to sand size. This trough has an open top and in the flow of the lead along the entire 60-ft. length of spout there is only a loss of 50° in temperature.

Table 1.—*Yearly Analysis of Bunker Hill Bullion*

	Au, Ounce	Ag, Ounce	Cu, Per Cent.	Sb, Per Cent.	As, Per Cent.	Bi, Per Cent.	Fe, Per Cent.
Before drossing	0.48	97.0	0.78	1.19	0.16	0.0012	0.045
After drossing	0.48	96.7	0.04	1.17	0.10	0.0012	0.008

The softening furnaces at Bunker Hill are comparatively small in comparison with those used in other plants, being of 110 tons capacity. Softening time required on average grade of bullion is 12 hr. The softened bullion analysis is as follows: Au, 0.50 oz.; Ag, 100.9 oz.; Cu, 0.03 per cent.; Sb, 0.12 per cent.

Skimming of the softeners is done cold after the furnace has been recharged with new bullion. The amount of skim taken off is 3.5 per cent. of the original bullion charged, and has the following analysis: Au, trace; Ag, 5.8 oz.; Pb, 64.8 per cent.; Sb, 18.2 per cent.; As, 1.40 per cent.

Desilverizing is done in four kettles of 105 tons capacity. The two center kettles are designated as gold kettles and the outer two as silver kettles.

The bullion, red hot, is tapped to the center kettles, where, after the litharge is removed, which has formed on the spout and on the kettle while filling, it is zinced for the first, or gold crust.

Refining Cycle

At the Broken Hill plant, the practice, followed almost identically by the Bunker Hill, is as follows: Five kettles of bullion are used to complete a cycle. Only the crust or skim on the fifth kettle is pressed and goes to the retorts, while that obtained on the first four is blocked and added to the next kettle, together with new zinc. The zinc schedule used for gold zincing is given in Table 2.

The gold kettle is worked after zincing as though it were a gold-silver operation and after the surface of the kettle is frozen the kettle is reheated, but here the Bunker Hill practice differs slightly from the Port Pirie practice, in not completing the desilverization in the same

kettle. The incompletely desilverized bullion is pumped to the outer kettles, because it is possible to keep the gold content in the silver skim considerably lower by this method of operation than by completing the silver operation in the same kettle.

TABLE 2.—*Zinc Used to Obtain Gold Crust*

For Kettle	Zinc per Ton Bullion, Pounds	Blocks from Kettle
No. 1	6.5	
No. 2	6.0	No. 1
No. 3	5.7	No. 2
No. 4	5.4	No. 3
No. 5	6.5	No. 4

Very little attention is paid to temperatures in desilverizing; excess lead carried into the silver-zinc-lead alloy does not cause any trouble, as the subsequent liquation process readily and economically eliminates the excess lead. The desilverizing operation is carried on as is usual in the art.

On the grade of bullion given above, the gold crust produced is 0.80 per cent. of the original bullion and the silver crust 2.5 per cent.

The saturation zinc and the antimony left in the softened bullion are eliminated in a reverberatory, or refining furnace, by bringing the temperature up to 1200° F., when the heat is cut off and air is blown into the bath. The skim from this furnace amounts to 3.4 per cent. of the original bullion, an average analysis being: Ag, 0.40 oz.; Pb, 70.3 per cent.; Zn, 11.3 per cent.; Sb, 4.6 per cent.

Under normal conditions the refiner skim is recharged to the lead blast furnaces. This practice is bad, but there is no alternative, except in the autumn, when an annual antimonial lead run is made, charging the accumulation of the year.

It is the writer's opinion that the ideal way to treat the desilverized bullion is by the Betterton chlorine process, wherein the saturation zinc is removed by the use of chlorine, but the difficulty at this plant is in finding a market for the zinc chloride produced.

All the lead produced by the Bunker Hill is of corroding grade; 75 per cent. is sold to corroders and storage-battery manufacturers. A yearly average analysis is given in Table 3.

TABLE 3.—*Yearly Average Analysis of Bunker Hill Lead*

Pb, Per Cent.	Cu, Per Cent.	Fe, Per Cent.	Zn, Per Cent.	Sb, Per Cent.	As, Per Cent.	Bi, Per Cent.	Ag, Per Cent.
99.9926	0.00027	0.0012	0.00136	0.0028	0.00019	0.00136	0.00027

Formation and Treatment of Crusts

By removing the gold and silver from the bullion, two types of skims, or crusts are made. The first, gold crust, contains practically all the gold and approximately 13 per cent. of the silver contained in the bullion, while the second, silver crust, contains all the remaining silver and a trace of gold.

The gold skim is dirty, because it picks up all the copper remaining after drossing and softening and also all the lead oxides remaining on the kettles after skimming subsequent to tapping from the softeners. Also, there is an excessive amount of zinc oxide present, due to the fact that only one pressing is made from five kettles. Fortunately, however, there is only a small amount of this type of crust made, and all the deleterious matter has been removed from the greater and more important silver crust.

No attempt is made to liquate the gold crust. It goes direct to the gold retort, from which the following retort bullion is obtained: Au, 41.94 oz.; Ag, 1321.5 oz.; Zn, 1.24 per cent. The gold retort bullion is charged to the gold cupel, to which are added the retort dross and all sweepings. The doré bullion from this cupel analysis is: Au, 25.7 fine; Ag, 965 fine. This is treated with sulfuric acid in the usual manner. Pure gold is sold to the United States Mint; silver 999 + fine is marketed to San Francisco, and the copper sulfate produced is used in the Bunker Hill concentrating mills.

The silver-zinc-lead alloy, for which the Bunker Hill is indebted to its Australian friends, has worked beautifully. This crust is charged into a kettle 18 in. dia. by 60 in. deep, made in two sections, a top half and a bottom half, joined together by bolts through flanges in which, in a groove, is a ½-in. water pipe to form a seal. The lower portion of the kettle is cast so that a 2-in. pipe can be threaded in and brought up high enough to act as a siphon for the liquated lead. (See Fig. 1.)

When the kettles are first charged, the lower portion is filled with desilverized lead, on to which the silver-zinc-skim is charged. Heat is applied to the top section only and by the time the charge has melted liquation has taken place. The lead separates from the alloy, sinks to the bottom portion of the kettle and runs out through the siphon pipe. This lead is practically free of silver and is returned to the desilverizing kettles shortly before they are pumped to the refining furnaces. In this way, 48 per cent. of the charge is recovered and goes with only a little additional treatment to refined lead, whereas formerly there was a large amount of work necessary to again get this lead to the refined state.

The liquated crust is dipped out of the kettle, poured for convenience on a water-cooled pan, and is then ready for the silver retorts. A charge to the liquating kettle is 1500 lb., which is melted down and skimmed in 2 hr., so that about 9 tons of crust go through the kettle in 24 hr. The

upper half of the kettle will treat 200 tons before it is burned out, and the lower half lasts indefinitely.

The liquated crust has the appearance of zinc metal and analyzes 48 per cent. zinc, 29 per cent. lead and 23 per cent. silver.

The average retort bullion produced will analyze: Au, 0.10 oz.; Ag, 12,800 oz.; Cu, 0.10 per cent.; Zn, 2.0 per cent.; whether made from 60 oz. or 250 oz. silver content in the lead bullion.

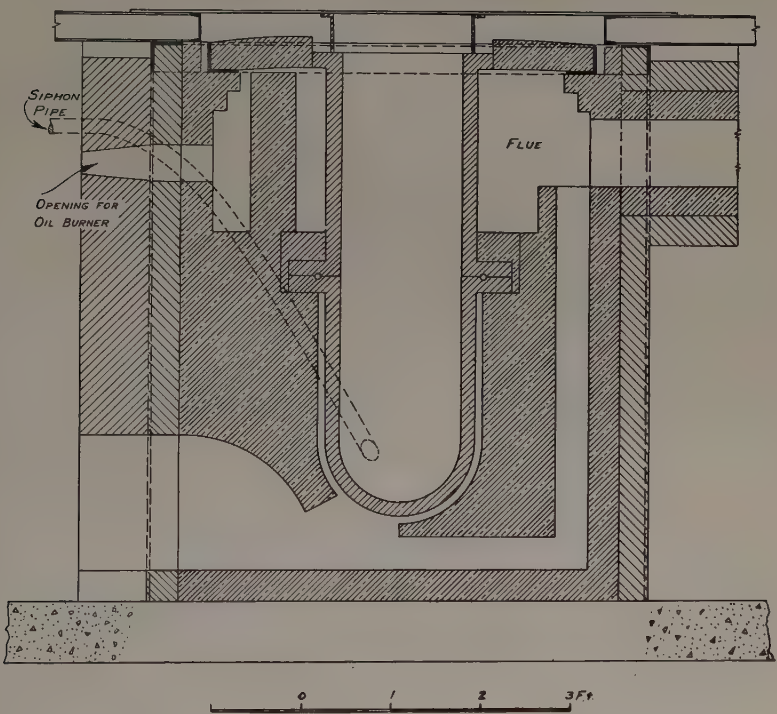

Fig. 1.

The average time consumed in retorting a 1200-lb. charge of high-grade crust is 6 hr. from charging to charging. The rapidity with which the zinc is condensed and a charge so high in zinc can be finished is due to an ingenious device which the superintendent of the silver refinery, A. Donaldson, invented—the placing of a hollow tile through the condenser, permitting cold air to pass through the center of the condenser and thus greatly adding to the cooling effectiveness.

Production of Metals

During the late summer and fall months of the year, the production of silver is 1,200,000 oz. per month, which is all treated in one gold and three

silver retorts. It is doubtful whether this figure is equalled by any refinery in the world.

The silver retort metal is treated in a silver cupel, Rhodes type, similar in size to the gold cupel. The resulting doré assays: Au, 0.015 fine; Ag, 997.4 fine. This is further treated in a Monarch melting furnace, where it is brought to 999+ fineness.

The gold value in the fine silver is only a few cents per thousand ounces of fine silver produced, which surely is an economical parting cost.

The copper dross obtained on the drossing kettles, amounting to 7.9 per cent. of the blast-furnace bullion, is treated in a reverberatory furnace 12 by 21 ft. by 26 in. deep, with a sloping bottom. A lead well is on the side at the fire end and a slotted slag and matte tap is in the middle of the furnace on the side.

Copper dross is accumulated during the month and the furnace is charged with this material toward the latter part of the month. To the charge is added siliceous ore and lime rock, which acts as a scorifier. In addition to the bullion produced in this furnace, amounting to 63.2 per cent. of the charge, a copper-lead speiss (21.9 per cent. of the charge) is produced of the following analysis: Ag, 95.0 oz.; Cu, 53 per cent.; Pb, 19 per cent.; Sb, 2.5 per cent.; As, 6.5 per cent. The speiss contains all the recovered copper that enters the smelter, as no matte is made in the blast furnace.

One other product is made in this furnace; namely, a copper-dross slag of the following analysis: Ag, 4.0 oz.; Cu, 2.5 per cent.; Pb, 26.6 per cent.; Sb, 5.0 per cent.; As, 2.3 per cent.; Fe, 13 per cent.; SiO_2, 17 per cent.; CaO, 3.5 per cent.; Zn, 6.5 per cent. As the copper speiss must be marketed at a copper plant where only a small portion of the lead content is paid for, of course the lead content of the speiss is kept as low as possible. Consequently, the mission of the copper-dross slag is to pick up as much lead as possible, and it is finally smelted in the lead blast furnaces.

After the copper-dross campaign has been finished, the same furnace is used to treat the skim from the softening furnaces. To one ton of this skim is added a small portion of high-grade galena ore low in silver together with coke breeze. From this charge there is produced bullion amounting to 56.5 per cent. of the charge, assaying: Ag, 15.0 oz.; Cu, 0.50 per cent.; Sb, 3.0 per cent.; As, 1.5 per cent.; also an antimonial slag amounting to 32.2 per cent. of the charge, analyzing: Ag, 0.5 oz.; Cu, 0.20 per cent.; Sb, 26.1 per cent.; Pb, 40.8 per cent.; As, 1.9 per cent.

Antimonial lead is produced only once a year at the Bunker Hill plant, because no cupola furnace has been constructed for this purpose. Antimonial slag is accumulated for 12 months; then there is a furnace campaign of approximately 9 days, smelting the entire accumulation in one of the lead blast furnaces, which is 48 by 180 in. at the tuyeres. The antimony content is kept as high as the trade will take it, and the

finished product will analyze: Pb, 80.46 per cent.; Sb, 19.16; Cu, 0.25; As, 0.12.

Arsenic is eliminated by the use of scrap iron in the blast-furnace charge and copper by cooling the lead in the well-known manner.

Scale of Operations at Bunker Hill

The following figures refer to operations at Bunker Hill:

Refined lead produced per day per man employed in lead and silver refineries	4.2 tons
Average life of dross kettles	3½ years
Average life of desilverizing kettles	540 heats
Number of retort hours per 1000 oz. gold and silver produced	1.8
Litharge made per 1000 oz. of gold and silver produced	325 lb.
Zinc condensed per hour in the silver retorts	95 lb.
Coal used per ton of refined lead produced, including complete refining of lead, gold, and silver	220 lb.
Oil used per ton of refined lead produced, including complete refining of lead, gold, and silver	5.0 gal.
Average life of retorts	60 heats

Acknowledgments

The author wishes to acknowledge the help and cooperation received in preparing this paper, from his assistants, J. B. Schuettenhelm, P. C. Feddersen and A. Donaldson.

DISCUSSION

G. E. Johnson, East Chicago, Ind. (written discussion).—The practice of producing a separate gold and silver crust is without doubt advantageous for certain lead refineries. The parting of doré is a necessary step in copper-refining practice, and where the doré from a lead-refining department of the same company can be parted in conjunction with the copper-refinery doré, it is a question whether the additional facilities required for separation of the gold and silver crusts at the lead refinery are justified.

Through the courtesy of Mr. Donaldson, of the Bunker Hill staff, we arranged to try out the Donaldson condenser. Our tests indicated a greater rate of cooling in the Donaldson condenser than in our standard condensers, made from old retorts. In fact, in some cases, perhaps due to the inexperience of our workmen, there was excessive cooling in the Donaldson condenser, with the consequent freezing of zinc in the bottom of the condenser. Our rate of zinc distillation, during the time interval from the first condensation of zinc to the completion of the distillation, averages 45 lb. per hour, when retorting zinc dross containing 25 per cent. zinc. From the information given by Mr. Beasley, the corresponding rate of distillation at Bunker Hill is somewhere between 95 and 120 lb. zinc per hour. In another test, we distilled 900 lb. of electrolytic zinc, and found the rate of distillation to be 53 lb. per hour.

We believe that the rate of distillation and condensation represents a balance between the rate of heat input to the retort charge and the condenser cooling capacity. As the heat input to the charge increases, the condenser capacity should be increased. High heat input requires a high furnace temperature. As the furnace temperature

increases, the deterioration of the retort and furnace linings increases rapidly. By increasing the furnace temperature, we have reduced the time of retorting a 1200-lb. charge from a normal of 6 hr. to 4 hr. With the 4-hr. retorting operation, we experienced a material increase in furnace and retort deterioration. We have concluded that a 6-hr. retorting period represents the most economical method of operation.

We use industrial gas as a fuel for the retorts. This gives a uniform heat; consequently we are able to determine the efficiency of retorting much more accurately than we could when we used fuel oil.

We consider 60 charges per retort a fair average life; however, we remove a retort from service whenever the rate of distillation decreases below normal. Our average number of charges per retort does not necessarily represent the ultimate life of a retort. In one case, disregarding the efficiency of zinc recovery, we realized a total of 198 charges.

Copper is eliminated at Bunker Hill by the production of a copper speiss, whereas at our plant copper is eliminated by the production of a copper-lead matte. The difference in the practice of the two plants is due to the difference in the outlets for the copper-bearing products.

As stated by Mr. Beasley, the copper speiss is sold to a copper smelter, where only a small return is realized on the lead content. Therefore it is desirable to reduce the lead content to a minimum. In our case, the copper-lead matte is shipped to our Tooele lead and copper smelter, where a similar copper-lead matte of their own production is treated. The major portion of the bullion received at our plant is decopperized at Tooele to a copper content of 0.008 per cent. The small quantity of copper received reduces the importance of the problem of copper elimination.

B. L. Brewer, Perth Amboy, N. J.—The two-crust formation as practiced at the Bunker Hill smelter is interesting; also the process of liquation. This differs from our Perth Amboy practice in that we produce only one crust and we do not at present liquate our crust. Even without liquating, and producing only one crust on the desilverizing kettles, we make the same amount of litharge per thousand ounces of doré as at Bunker Hill; that is, 325 lb. of litharge per thousand ounces of doré.

One of the primary objects of liquation is reduction of litharge production, and I wonder why Bunker Hill cannot produce less litharge per thousand ounces of doré than we do at Perth Amboy. Also, why cannot they liquate the gold crust?

T. D. Jones, Perth Amboy, N. J.—Mr. Beasley says that temperature control is not used in the Bunker Hill practice. We wonder if there is not a distinct advantage in having temperature control in the desilverizing process; in other words, if part of the liquation can be brought about in regular desilverization instead of sending a high-lead dross to the retorts. We maintain strict temperature control at Perth Amboy and believe it is one of the prime factors in regulating concentration.

We check Mr. Johnson very closely on distillation of zinc. I have tried for eight or nine years to get the rate of zinc distillation up, but we average only about 55 lb. of zinc per retort-hour. In other words, we feel as Mr. Johnson does, that it is the amount of heat in the retort setting which governs the rate of distillation. We have never been able to get 95 lb. of zinc per hour.

Another question concerns the refining of doré. We must have a certain lead content in the retort metal, in order to refine doré; in other words, if you liquate too far in the liquator, you will reduce the lead content of the retort metal to such an extent that there will not be enough lead present to refine the doré. If that is done, lead has to be added, in one form or another. We feel that retort metal has to carry a certain amount of lead in order to be able to refine doré. If it does not, the lead has to be added in our case, which is an expensive proposition.

F. F. Colcord, New York, N. Y.—Does not the practice of taking off the gold crust remove the impurities, so that when the silver is cupeled there is less impurity to be removed?

T. D. Jones.—That is right.

F. F. Colcord.—I have a communication in which the question is asked whether the gold crust can be liquated as well as the silver crust. In other words, does the impure gold crust liquate like the pure zinc silver crust. Can you tell us anything on that, Mr. Johnson?

G. E. Johnson.—Only what I have heard indirectly—that the impurities present in the gold crust largely interfere with its ability to liquate. I believe the gold crust can be liquated, but more zinc is oxidized in the liquation. It is a matter of temperature.

B. M. O'Harra, Maurer, N. J.—As Mr. Brewer said, we are not using liquation at Perth Amboy now, but we have done some experimental work along that line. As you have suggested, it is much more difficult to liquate crust which has not been purified by removing a previous gold crust, particularly when the crust is already fairly high in silver. The higher silver content raises the melting point of the rich alloy considerably.

We have found that the difficulty can be overcome by using a cover slag; for example, zinc chloride, which fuses at a low temperature, or a mixture of calcium and sodium chlorides, will form a slag that will dissolve the oxides and some of the other impurities and permit a good liquation. By this means we have been able to get a lower lead content than at Bunker Hill, obtaining high-grade alloy containing as low as 5 per cent. lead, but we produce a slag which must be treated in some way afterwards and, as Mr. Jones says, the alloy can hardly be cupeled by itself without having more lead in it.

C. R. Hayward, Cambridge, Mass.—As one who has something to do with the teaching of metallurgy, this interests me. I always tell my students that the abandonment of a process is no reason why we should not review it and possibly get a successful application of it in the light of progress in metallurgy. Along similar lines with this, although on a different subject, I have been interested recently to learn that at Clifton, Ariz., they have gone back to trying the old process in copper converting, in which a small amount of blister is first made to take out the precious metals and then the remainder of the copper is converted and does not require electrolytic refining. I merely take this occasion to pass along the thought that many processes which were in use years ago along various metallurgical lines are worthy of renewed inspection, because we have a great deal more metallurgical knowledge than when they were used, and sometimes just a little readjustment might make some of them of value in specific cases today.

F. F. Colcord.—The point that strikes me in this paper is the economic one of the saving of the parting cost on 87 per cent. of the silver. Mr. Johnson has touched on that.

A. F. Beasley.—Mr. Johnson is correct when he says that the rate of distillation depends on the heat input into the retort, but this, of course, is related to the rapidity of condensing the zinc vapor therefrom. I believe we have all seen condensers moved away from the retorts on account of the pressure set up therein. The Donaldson condenser is not too large to cause freezing of the zinc when properly operated, and gives, on account of its construction, a much greater cooling rate. I believe most of us know what happens when we heat a retort and have to hold the fire after the charge has become hot. I am sure, from our experience, that the rate of distillation is

not directly in proportion to the heat applied, but that the hotter the fire, the greater the rate of distillation.

Regarding retort furnace repairs, Mr. Johnson is again correct. Many years ago we corrected this by feeding fuel at the top of the retort, causing the flame to impinge against the bottom, which in our case is a hot coal-ash slag. By introducing the fire at this point, there is not an excessive amount of deterioration of the brick work on the sides or top of the furnace. It has never been necessary at this plant, to my knowledge, to discard a retort from service because of the inability to condense zinc therefrom in the normal length of time.

Replying to Mr. Brewer and Mr. Jones, I might say that the making of two types of crusts at the Bunker Hill smelter has proved a very economical practice. When this plant was constructed it was thought there would not be a sufficient intake of silver to make it economical to install electrolytic parting, consequently the sulfuric acid method was adopted, and has been continued since.

In using this method, we found that the cost of steam for boiling and evaporating was a very large item, first because of a high coal cost, and second because in no other part of the plant is steam used. Consequently, the entire cost of the operation of the boiler must be charged against the parting operation.

It seems to me that it is evident that when one can reduce the amount of parting from 100 per cent. to 13 per cent. through the use of a selective separation in the desilverizing kettles, the process should be given considerable consideration.

We wonder whether Perth Amboy does not do as Mr. Johnson does—pass the doré along to a copper refinery for parting, where undoubtedly, due to a very large silver output, the cost is at a minimum. We feel that anyone with conditions similar to ours cannot help but see the advantage of the two-crust operation.

Regarding liquation, naturally, due to the five-kettle cycle operation in removing the gold, the crust contains a much greater proportion of zinc and lead oxides than if the crust were pressed and liquated from one operation. Similarly, the copper from the five-kettle cycle is concentrated in the one gold crust. We have liquated gold crust, but on account of its dirty condition there is no advantage over retorting, although it is about a stand-off.

The litharge figure of 325 lb. per thousand ounces of gold and silver produced contains not only the weight of the litharge but also the weight of all clean-up, including cupel bottoms, broken retort bottles and melting furnace bottoms. Straight litharge figures for the past two years, 1928 and 1929, without foreign material added, are:

	Litharge Made, Lb.	Silver Produced, Oz.	Litharge per 1000 Oz. Silver, Lb.
1928	2,317,108	7,634,674	304
1929	2,415,304	8,614,398	281

Previous to the time of liquation at this plant, our average retort metal assayed 3500 oz. per ton, and 530 lb. of litharge was made per thousand ounces of silver produced, so it can readily be seen that our practice has been greatly improved by the use of the liquating kettle.

Possibly I did not make myself sufficiently clear regarding temperature control. Like everyone else, we heat our kettles to the point where the returns will melt within a certain period, but we do not try to liquate the silver skim in the desilverizing kettle, for the very good reason that we have found it much cheaper to accomplish this same result in the liquating kettle.

We have never found it necessary to add lead in cupeling, our retort metal assaying 13,000 oz. per ton. I have no doubt that Mr. Jones is correct in his statement, but it does not hold for 13,000-oz. retort metal. If the zinc in our retort metal were higher, we might have to do this, but not with the grade we produce.

Investigation of Anodes for Production of Electrolytic Zinc

By H. R. Hanley,* C. Y. Clayton† and D. F. Walsh‡, Rolla, Mo.

(New York Meeting, February, 1930)

Electrolytic zinc produced from sulfate solution and with pure lead anodes is always contaminated with a small and varying percentage of lead. The purpose of this investigation is to determine the characteristics of several lead alloys and their influence on the products of electrolysis and power consumption.

The study of alloyed anodes has been a matter of interest for some years. Tainton, Taylor and Ehrlinger prepared a highly interesting paper[1] on this subject a year ago, in which the effects of silver-lead and some ternary alloys of lead-silver-tin were described, and the presentation of any paper on insoluble anodes will owe much to the inspirational results achieved by Dr. Colin G. Fink and Li Chi Pan in their work on insoluble anodes for the electrolysis of brine,[2] and to other published papers by Dr. Fink and his coworkers embracing the development of the copper-silicon-manganese anode for the production of electrolytic copper and the general subject of anode development. The present paper describes a study of various combinations of 10 metals. The general plan of the work provides for the fabrication of the alloyed anode, its adequate conditioning by preparatory electrolytic operation, the production of cathode zinc and its analysis, the determination of the anode polarization, and other features.

A list of the anodes tested, showing composition and polarization at various current densities together with the lead assay of the cathode zinc, is shown in Table 1.

Determination of Anode Polarization

In the determination of anode polarization under operating conditions it is essential that the anode area used be restricted to the side facing the cathode, in order that these measurements may be referred to a definite current per unit of area. It becomes necessary, therefore, to insulate the inactive face of the anode, to prevent the leakage of lines of force

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[1] U. C. Tainton, A. G. Taylor and H. P. Ehrlinger: Lead Alloys for Anodes in Electrolytic Production of Zinc of High Purity. *Trans.* A. I. M. E. (1929), 192.

[2] C. G. Fink and Li Chi Pan: Insoluble Anodes for the Electrolysis of Brine. *Trans.* Amer. Electrochem. Soc. (1926) **49,** 85–134.

around the edges where they would spread over an increased area. This would give an erroneous relation of polarization to true current density.

The anodes of lead alloy were made in pairs having the inactive face insulated with a suitable varnish. Each pair was conditioned by electrolysis for several hours, to develop uniform surface condition. Fresh

TABLE 1.—*Anode Composition and Polarization and Lead Content of Cathode Zinc*

Anode No.	Intended Composition, Per Cent.											Anode Polarization, Current Density, Amp. per Sq. Ft.		Decrease in Potential below Lead at Current Density, Amp. per Sq. Ft.		Per Cent. Pb in Cathode Zn at Current Density, Amp. per. Sq. Ft.	
	Pb	Ag	As	Ca	Ba	Mg	Tl	Cd	Hg	Al	Bi	50	100	50	100	100	30
A	100.00											0.500	0.512			0.064	
1	97.5	2.5										0.421	0.447	0.079	0.065	0.010	
2	99.0	1.0										0.415	0.440	0.085	0.072	0.014	
3	99.5	0.5										0.442	0.458	0.058	0.054	0.029	
4	96.7		3.3													0.107	
5	99.0		1.0													0.137	
6	99.5		0.5													0.142	
7	98.0	1.0	1.0									0.407	0.423	0.093	0.089	0.0126	
8	99.0	0.5	0.5									0.469	0.490	0.031	0.022	0.0160	
9	98.8	1.0	0.2									0.350	0.370	0.150	0.142	0.020	
25	97.38			0.1	2.52												
26	97.62			2.28								0.260	0.281	0.240	0.231	0.043	
27	99.95			0.05								0.393	0.405	0.107	0.107	0.104	
28	96.72	1.0		2.28									0.280		0.232	0.010	
29	97.86	1.0		1.14									0.270		0.242	0.009	
30	98.46	1.0		0.54									0.231		0.281	0.014	
10	98.0								2.0			0.336	0.500	0.164	0.060	0.331	
11	98.0										2.0	0.370	0.415	0.130	0.097	0.028	
12	98.0					2.0						0.400	0.415	0.140	0.097	0.060	
13	98.0							2.0				0.445	0.458	0.055	0.064	0.127	
21	95.0						5.0					0.416	0.430	0.084	0.083	0.012	
14	99.5						0.5										0.120
15	99.0						1.0										0.090
16	98.0						2.0										0.070
17	97.0						3.0										0.030
18	96.0						4.0										0.003
19	95.0						5.0										0.002

aluminum cathodes were then inserted and the operation continued for 11 hr. at a current density of 100 amp. per sq. ft. The cathode zinc was then stripped from the aluminum sheet and analyzed.

The electrolyte consisted of acid zinc sulfate solution containing approximately 68 g. Zn per l. and 200 g. H_2SO_4 per l. initially and approximately 30 g. Zn per l. and 257 g. H_2SO_4 per l. at the end of each cycle. Many pairs of anodes were subjected to electrolysis in the presence of manganese sulfate and also with this salt practically absent, in order to observe the specific effect of the oxidation product of manganese on limiting the anode corrosion.

The anode polarization measurements were made as follows: One anode and one cathode were placed in the grooves of a Haring cell and current applied at a current density of 50 amp. and also 100 amp. per sq. ft. Behind the active anode the other anode was placed but not connected in the electrical circuit. Potentiometric readings were then taken between the polarized anode and an inactive one at each of the current densities referred to above. These readings are not uniform until after about one hour of operation in the testing cell, when they become reasonably constant. A fresh portion of solution was used in the test cell for each pair of anodes, containing approximately 78 g. Zn per l. and 182 g. H_2SO_4 per liter.

Factors in Electrolytic Production of Metal

In the production of a metal electrolytically three factors influencing power consumption must be considered: *viz.*, anode potential, potential due to ohmic resistance and cathode potential, which together give the terminal voltage of the cell. The potential due to ohmic resistance is fixed by the spacing of electrodes, acidity and temperature of the electrolyte, therefore after the most suitable relations are established this component is practically constant. The cathode potential for zinc involving hydrogen overvoltage is practically fixed, for a solution of definite purity, by controlling the current density and temperature. A high value for this component is always desirable. The anode potential involving oxygen overvoltage is influenced mainly by the composition of the anode and current density. Inasmuch as the latter is fixed for a given plan of operation, the chief variation in the anode potential may be brought about by a variation in anode composition.

Results of Investigations

Investigations of this kind are seldom completed with finality because the possible quantitative and qualitative combinations of metals are unlimited.

Mr. Tainton showed the benefit of the silver-lead anode in lowering the electromotive force of the cell and inhibiting lead in the cathode zinc to a remarkable degree.

Among the 28 different anodes tested by the authors, the calcium-lead alloys were the only ones to exhibit remarkable lowering of the anode potential, with the exception of the silver-arsenic-lead anode devised by Mr. Tainton. The calcium-lead anode, however, does not possess sufficient passivity to prevent transfer of some lead to the cathode zinc. The passivity is developed, however, if 1 per cent. silver is introduced into this binary alloy. Then it becomes equal to the silver-lead anode in this respect, with the enhanced value of much greater lowering of the anode

potential than any other anode tested. This lowering amounted to from 0.20 to 0.27 volt, or from 40 to 50 per cent. below the potential of the pure lead anode.

Anodes, generally, corrode at a greater rate with a higher current density than with a lower one. Therefore the effect of a high current density in this respect shortens the time of a test to yield a positive result.

Thallium in lead has practically no effect on passivity of the anode from 0.5 to 2.0 per cent., but beyond this concentration there is a decided effect in the lowering of the lead in the cathode zinc. For example, with a low current density (30 amp. per sq. ft.) for the entire Tl series, the following figures present this relationship:

Thallium in Lead Anode, Per Cent.	Lead in Cathode, Per Cent.
0.5	0.120
1.0	0.090
2.0	0.070
3.0	0.030
4.0	0.003
5.0	0.002

A higher percentage of thallium up to 15 per cent. does not show any improvement over the alloy containing 5 per cent. The lead anode containing 5 per cent. thallium, when operated at 100 amp. per sq. ft., produces a cathode zinc containing 0.012 per cent. Pb.

The lead content of the cathodes shown in Table 1 are of interest only in a relative sense. There were no agents added to the electrolyte to contribute to the anode passivity, therefore these figures are not intended to represent a specific lead assay of the cathode but values comparable with one another. Addition agents that contribute to inhibiting the anode corrosion probably should affect these values in a similar manner.

Manganese in the electrolyte lends a degree of passivity to the anode. Table 2 shows this effect, by the lessening of lead in the cathode zinc.

Summary

There were 28 lead-alloy anodes tested in respect of passivity and anode potential.[3] Outside of the silver-lead anode devised by U. C. Tainton, the only series that showed outstanding characteristics were those containing calcium lead and thallium lead. The calcium-lead anode exhibited remarkable lowering of the anode potential, amounting to approximately 50 per cent. below the potential of pure lead. This alloy did not exhibit passivity better than pure lead, but it became passive when 1 per cent. silver was added. Then it became equal to the silver-

[3] The lead-calcium and lead-barium alloys were furnished by the Midwest Carbide Co. of Keokuk, Iowa.

lead in this respect, with the enhanced value of greater lowering of the anode potential than any other anode tested. Thallium over 4 per cent. was shown to exhibit remarkable passivity but with very little lowering in the anode potential. The influence of manganese on stabilization of alloyed anodes was also determined.

TABLE 2.—*Effect of Manganese in Solution in Lessening Lead in Cathode Zinc*

Anode No.	Intended Anode Composition, Per Cent.					Manganese in Solution, Grams per Liter	Per Cent. Lead in Cathode Zinc	Current Density, Amp. per Sq. Ft.
	Pb	Ag	As	Ca	Tl			
A	100					Tr	0.190	100
A	100					0.6	0.064	100
1	97.5	2.5				Tr	0.020	100
1	97.5	2.5				0.6	0.010	100
2	99	1.0				Tr	0.027	100
2	99	1.0				0.6	0.014	100
3	99.5	0.5				Tr	0.043	100
3	99.5	0.5				0.6	0.029	100
7	98.0	1.0	1.0			Tr	0.025	100
7	98.0	1.0	1.0			0.6	0.126	100
8	99.0	0.5	0.5			Tr	0.038	100
8	99	0.5	0.5			0.6	0.016	100
9	98.8	1.0	0.2			Tr	0.044	100
9	98.8	1.0	0.2			0.6	0.020	100
269	97.62			2.28		Tr	0.087	100
269	97.62			2.28		0.6	0.043	100
WEC	99.95			0.05		Tr	0.319	100
WEC	99.95			0.05		0.6	0.104	100
17	97.00				3.0	Tr	0.060	30
17	97.00				3.0	0.6	0.030	30
18	96.00				4.0	Tr	0.010	30
18	96.00				4.0	0.6	0.003	30
19	95.00				5.0	Tr	0.003	30
19	95.00				5.0	0.6	0.002	30

APPENDIX

The alloys prepared for this investigation fall into four groups according to their mutual solid solubility and their tendency to form compounds.

A. No compounds form.

1. There is a eutectic.
   a. Metals are insoluble in the solid.
   Lead-silver............ 414
   Lead-arsenic........... 414

2. No eutectic forms.
   a. Metals are partially soluble in the solid.
   Lead-thallium....... 416

1 2 3 4 5 6

Fig. 1.—Anode 13. Pb, 98; Cd, 2 per cent.
Fig. 2.—Anode 30. Pb, 98.46; Ca, 0.54; Ag, 1 per cent.
Fig. 3.—Anode 28. Pb, 96.72; Ca, 2.28; Ag, 1 per cent.
Fig. 4.—Anode 29. Pb, 97.86; Ca, 1.14; Ag, 1 per cent.
Fig. 5.—Anode 9. Pb, 98.8; Ag, 1; As, 0.2 per cent.
Fig. 6.—Anode 26. Pb, 97.62; Ca, 2.28 per cent.
Original magnification 200. Reduced one-half.

b. Metals are partially soluble in the solid.
Lead-cadmium (4%)... 414
Lead-mercury (35%)... 414
Lead-bismuth (37%)... 414
Lead-magnesium (3.3%) 415

B. Compounds form.
1. There is a eutectic.
a. Metals are insoluble at least on the lead side.
Lead-calcium........ 414

The figures in parenthesis indicate the percentages of lead soluble at the eutectic melting temperature. The number following each alloy is the page in International Critical Tables, Vol. 2, wherein there appears an equilibrium diagram.

The lead-calcium diagram indicates that the alloy used would consist of lead associated with a compound Pb_3Ca.

Through the courtesy of the metallography laboratory of the Hawthorne Works of the Western Electric Co., photomicrographs were prepared for each alloy investigated. Anodes 1 to 9 showed lead associated with certain amounts of eutectics. Anodes 11 to 19 and anode 27 were solid solutions, in some cases showing a precipitation of one phase. Six of the photomicrographs are shown in Figs. 1 to 6

The most interesting points brought out by a microscopic study of the anodes is the great difference in structure between anodes:

26 = 2.28 per cent. Ca,
28 = 2.28 per cent. Ca, 1 per cent. Ag
29 = 1.14 per cent. Ca, 1 per cent. Ag
30 = 0.54 per cent. Ca, 1 per cent. Ag

No attempt will be made at this time to show any relation between structure and anode potential. Attention is directed to the fact that anode 30, with 0.5 calcium and 1 per cent. silver, gave the lowest potential value.

This work will be continued and a careful microscopic study of the alloys will be undertaken.

DISCUSSION

C. Y. Clayton.—We did very little from the microscopic standpoint with these particular alloys. We merely show these few photomicrographs to bring out especially the fact that the best anodes were not the true solid-solution type of anode. Anode 13, of lead and cadmium, is a true solid-solution type, and is the only one of this type that really gives good results. The main thing we want to bring out is the fact that there is a great difference particularly between anodes 26 and 28; 26 is one with calcium and lead, and in 28 we have introduced 1 per cent. of silver. Also, there is a great

difference between 28, 29 and 30, where we have decreased the calcium progressively, with the silver content the same. We have had no time to go into a thorough study of these from a microscopic standpoint and we simply offer them for inspection.

C. R. Hayward, Cambridge, Mass.—I am interested to know whether the authors made any experiments as to the method of casting these anodes. Some years ago in some work I did in connection with some lead-antimony alloys, I found that when they are cast flat in an iron mold, the face that was chilled in contact with the mold corroded much more slowly than the face that was upward and cooled more slowly. In making some microscopic studies, it was obvious that the chilling effect had affected the crystallization. The face that corroded more slowly showed the fine dendritic structure of some constituent just coming out of solution, whereas the part that corroded more rapidly showed the more perfect crystallization of a substance which had the opportunity to come out in its normal way. I feel that a final word on this subject of the corrodability of anodes must include the method by which they are cast.

C. Y. Clayton.—The anodes were cast in a vertical mold made up of two plates of graphite between which was a heavy iron wire to give the anode the proper shape. We were rather pressed for time and made no study of the effect of direction of crystals on the anode corrosion. The anodes were approximately ⅛ in. thick.

C. G. Fink, New York, N. Y. (written discussion).—For a number of years we have investigated the lead alloys as anodes. In 1921, Fink and Eldridge published their detailed findings on the lead-thallium series tested as anodes in sulfate solutions.[4] It was shown that, starting with either pure lead or pure thallium, the anodic solubility of the lead-thallium alloys dropped very sharply to about one-eighth the solubility of pure lead. Furthermore, the low-solubility section of the curve is remarkably long, extending over the range of about 35 to 75 per cent. lead. In that same paper the behavior in sulfate solutions of anodes of lead-bismuth, lead-tin, lead-tin-barium, lead-tin-bismuth-silver, thallium-tin-barium and others are referred to.

In 1924, Fink and Pan[5] reported upon results obtained with the lead-silver and lead-silver-manganese anodes, using sodium chloride solutions which are far more corrosive than the sulfate solutions. Here, too, it was found that, starting with either pure lead or pure silver, and adding small percentages of the other metal, the anodic solubility dropped very sharply down to but a very small fraction of the solubility of either pure metal. Likewise the electrode potential dropped very abruptly. Small percentages of manganese added to the Pb-Ag alloy anodes did not improve their performance, which is contrary to our observation in sulfate solutions.

Our researches on the lead alloys have been continued and we hope shortly to publish a fourth paper in the series.

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[4] C. G. Fink and C. H. Eldridge: Electrolytic Corrosion of Lead-thallium Alloys. *Trans.* Amer. Electrochem. Soc. (1921) **40**, 51–61. U. S. Patent 1384056 (July 12, 1921).

[5] C. G. Fink and Li Chi Pan: Insoluble Anodes for the Electrolysis of Brine. *Trans.* Amer. Electrochem. Soc. (1924) **46**, 349; (1926) **49**, 85. See also Fink and Lowe: U. S. Patent 1740291 (1929).

Improvements in the Metallurgy of Quicksilver

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(San Francisco Meeting, October, 1929)

During the war period of quicksilver activity there were a number of departures from what may be termed the classical quicksilver metallurgy. Attempts were made to beneficiate low-grade ores by gravity concentration and flotation; mechanical furnaces began to replace the Scott furnace and the vertical coarse-ore furnaces, and condensers of sewer tile and redwood tanks were used in place of the older brick and stone condensing chambers. Other developments that have occurred during the current period of quicksilver activity are mainly in the nature of improvement and further refinement of those started during the war period.

Each period of quicksilver activity has been accompanied by the proposal of various new processes, including wet methods for the treatment of quicksilver ores and innovations in furnaces, retorts and condenser equipment. As far as the author knows, nothing of economic significance in the way of "new processes" has been developed. There is in fact no real need for any essentially new process for the treatment of quicksilver ores; the direct furnace treatment is simple and inexpensive. Moreover, with rare exceptions, the quicksilver industry has no complex or refractory ore problems corresponding to those which are receiving more and more attention from metallurgists in other branches of the nonferrous field. Under these conditions improvements in quicksilver metallurgy must be looked for through the adaptation of current developments in engineering and metallurgy generally to the particular needs of quicksilver practice rather than through the invention of new processes.

Preliminary Treatment of Quicksilver Ores

At Sulphur Bank, in the treatment of old dump material left by the early operators, wet screening is being used as a preliminary step in concentration ahead of flotation. Power shovels and trucks are used to deliver some 400 tons of ore per day to the lower terminal of a hoist, which in turn discharges the material at the top of the screening plant. Wet screening is carried on in two stages. Depending on the character of the material, 1-in. or 2-in. punched screens are used on the first trommel.

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The oversize from this passes to a picking belt. The undersize is conveyed to twin trommels where two products are usually made. The −3-mm. material, in the form of pulp, passes directly to a storage tank ahead of the flotation plant. The +3-mm. material up to −½ in. or −1 in., depending upon the distribution of values, passes to a storage bin ahead of the ball mill.

With heads to the screening plant running approximately 0.1 per cent.—that is, 2 lb. mercury per ton—it has been found possible to recover 75 per cent. of the values with a 4 to 1 concentration; that is, 400 tons of 2-lb. material yields 100 tons of 4 to 5-lb. screenings. This practice is particularly applicable to a condition like that in the old dumps at Sulphur Bank, where the values occur chiefly in the fines and where the cost of delivering the material to the screening plant is low.

At another mine in California hand sorting is employed on a large scale to obtain a furnace grade of ore.

Flotation

The application of the flotation process to quicksilver ores and to old mercury-bearing dumps has received considerable attention within the past few years. During the early development of the flotation process a number of attempts were made to utilize flotation in quicksilver metallurgy but without notable success. One difficulty appears to have been the low grade of the concentrate produced. A low-grade cinnabar concentrate carrying a large proportion of gangue slime is not only difficult to filter and dry but also presents difficulties in its subsequent treatment for the recovery of the metal.

With our present-day knowledge of the flotation process much better results can be obtained and it may be safely stated that flotation has a definite place in quicksilver metallurgy. In the treatment of ores, cinnabar is the principal mineral to be recovered. In treating old dumps from amalgamation or quicksilver condenser products, finely flowered metallic mercury and synthetic mercuric sulfide are found. All of these substances are readily amenable to flotation.

Generally speaking, cinnabar floats with great readiness and good recoveries can be made from low-grade material. Laboratory tests on several different ores have yielded tailing running from 0.01 to 0.05 per cent.; that is, 0.2 to 1.0 lb. Hg per ton. As a specific example, the following results obtained on a sample assaying 0.35 per cent. Hg are quoted. The concentrate assayed 62.8 per cent. Hg and the tailing 0.01 per cent. Hg, which corresponds to a 97 per cent. recovery. The high grade of the concentrate is important, as a concentrate of this grade can be retorted easily for the recovery of the metal. Experience with actual mill practice indicates that these laboratory results can be duplicated on a

large scale. Aerofloat has been found particularly selective for cinnabar and a small amount of copper sulfate is sometimes helpful. Flotation equipment in which the froth is under close control is essential for the production of a high-grade concentrate. The Kraut cell exemplifies this type of equipment.

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FIG. 1.—GENERAL VIEW OF SUPLHUR BANK PLANT. (COURTESY J. G. PARMELEE.) Center, hoist, screening and flotation plants; left, tailing pile; right, furnace plant.

FIG. 2.—GENERAL VIEW OF SULPHUR BANK FURNACE PLANT LOOKING TOWARD CLEAR LAKE. (COURTESY J. G. PARMELEE.)

The Sulphur Bank operation presented a rather unique flotation problem. The old dump material was not only highly acid but contained considerable elemental sulfur, and there was a large amount of extremely fine slime resulting from the action of the acid on the gangue. The flotation plant equipment includes a Hardinge mill, a Dorr classifier,

Minerals Separation Sub A cells, Dorr thickeners and an Oliver filter; also a two-cell Kraut unit for cleaning concentrate.

Originally the −3-mm. pulp coming from the screening plant was delivered to a storage tank equipped with a Devereux agitator and thence directly to the classifier. The coarser material from the screening plant was delivered to a bin and thence to the Hardinge mill together with the oversize from the classifier. Later, it was found that better results could be obtained by routing the −3-mm. pulp through the ball mill. Apparently the passage of this pulp through the mill served to polish the cinnabar particles and release adhering gangue slime. The flotation tailing averaged about 0.05 per cent.—that is, 1 lb. mercury per ton—and with the Kraut cleaning unit a 35 to 50 per cent. Hg concentrate was made. With this grade of concentrate, thickening was unnecessary. The cleaner froth was delivered directly to plate filters, which were constructed from old 50-gal. oil drums with filter bottoms.

Part of the sulfur contained in the old dump material showed a marked tendency to float. It was found that this portion of the sulfur could be floated readily with a small amount of kerosene before the cinnabar was conditioned and floated. Some of the remaining sulfur entered the cinnabar concentrate but the greater portion was discharged with the tailing. When elemental sulfur is present its elimination is essential, as a cinnabar concentrate carrying a large proportion of elemental sulfur presents serious difficulties in subsequent treatment.

The extent to which flotation has a place in quicksilver metallurgy is a question of interest. Speaking generally, a good recovery and a high-grade concentrate can be obtained. The alternative to milling and retorting, of course, is the direct furnace treatment of the ore, which is both simple and inexpensive. The great majority of quicksilver operations are on a moderate scale, say 50 to 100 tons per day. Within this range the first cost of a flotation plant including retorts for treating the concentrate will not differ greatly from that of a furnace plant.

Milling has the advantage that the units are relatively mobile and can be moved to a new site when the first operation is at an end. Moreover, milling plant equipment in general has a fair salvage value. On the other hand, even with a mechanical furnace, the loss in moving a furnace plant will be considerable.

For furnace treatment, crushing to 1 to 2 in. is usually sufficient. Thus the power required for crushing is small and the other power requirements for a furnace plant are also low. The over-all cost of furnace operation need not exceed one to two dollars per ton. With milling the power for fine grinding may be a considerable item. Naturally no general rule can be laid down as to the relative merits of the two procedures. With ore running 0.25 per cent. Hg—that is, 5 lb. per ton and higher—direct furnace treatment, in general, will give a more

favorable ecomomic outcome. The best opportunities for flotation seem to lie in the treatment of relatively large tonnages of lower grade material where fine grindings is not an expensive item and where the costs of mining and transportation are low.

Furnace Treatment

During the last few years, mechanical furnaces, particularly the rotary, have been widely used. For many years the Scott furnace was the accepted type of fine-ore furnace and in the recent trend toward mechanical furnaces the merits of the Scott furnace have been more or less overlooked. Perhaps one reason for this is that Robert Scott, the inventor and builder of most of the Scott furnaces in this country, is no longer living and has left no direct successor. As far as metallurgical results are concerned, the Scott furnace leaves little to be desired. Its fuel requirements are no greater than the rotary and it has the advantages of requiring no power and having no moving parts that require mechanical attention and replacement. Even with dusty ores the Scott furnace produces little dust, whereas the mechanical furnaces, particularly the rotary, are dust makers.

It is commonly believed that the construction cost of the Scott furnace is higher than that of the rotary. The cost of a rotary is not so definitely fixed as that of the Scott furnace; the cost can be decreased by using a light-weight shell, omitting an insulating lining between the refractory lining and the shell, and in other ways. This may be an advantage for a short-lived operation but in the long run the economy of such practice is doubtful. With most ores the rotary kiln produces so much dust that special equipment is necessary for cleaning the gases before they enter the condenser system. The cost of the dust-collecting device is properly a part of the rotary kiln cost. The substitution of oil fuel for wood in the Scott furnace makes possible an increase in capacity of at least 25 per cent. over the old rating. When all of these factors are considered, there is little difference between the first cost of a Scott furnace and that of a rotary furnace plant. The choice of furnace necessarily depends on many local factors and any general dictum on this point is impossible.

Rotary Kiln

The external dimensions of rotary kilns now in use range from 40 to 70 ft. long and 3 to 5 ft. dia. The lining is from 4½ to 6½ in. thick, depending on whether or not an insulating layer is placed between the refractory lining and the shell. Monolithic linings have been used with success in a number of cases. The best combination for such a lining is crushed firebrick, 100-mesh firebrick dust, and a quick-hardening cement of the portland cement type, such as "Lumnite" cement. In

some cases roasted ore from the furnace has been used as the coarse aggregate. This lining is tamped into place back of movable forms. At Sulphur Bank, where the ore is soft, a lining of this type has been in service for two years with only minor repairs. With hard or abrasive ore the life may not exceed 4 to 6 months, but under such conditions a firebrick lining has shown no better life.

The feeding of ore into a rotary kiln of small diameter has been a difficult problem. The so-called "grasshopper" or reciprocating tube feeder has been extensively used. In order to avoid back feeding, it is necessary for this feeder to extend from 6 to 9 ft. within the kiln. At Sulphur Bank, back feeding has been avoided by placing a set of six helical blades just within the feed end of the kiln. This construction is similar to that commonly used in rotary dryer practice.

The inclination of the rotary kiln ranges from 0.5 to 0.75 in. per foot and the speed of rotation is usually from 1 to 2 r.p.m. The time of residence of the ore in the kiln is about one hour. The capacity of the rotary kiln ranges from 40 to 100 tons per day, depending on its size and the character of the ore. The moisture content of the ore is one of the important factors. At the Opalite plant of the Mercury Mining Syndicate a kiln 4 by 70 ft. is handling 90 to 100 tons of ore per day with a fuel consumption of slightly less than 7 gal. of oil per ton. This ore carries only a small amount of moisture but is particularly refractory because the cinnabar is disseminated through a siliceous sinter. The burning temperature for this ore is considerably higher than for the usual quicksilver ore. The feed for a rotary kiln is usually crushed to 2 in. An excess of fines tends to reduce furnace capacity.

Sufficient attention has not been given to the selection of oil burners for the rotary kiln. The best results are obtained with a burner that delivers a long luminous flame, which gives the best heat transference. This type of burner is exemplified by the equipment manufactured by the Ryder Engineering Co. That company's burner is of the pressure atomizing type with auxiliary low-pressure air for shaping the flame. The calcine is discharged into a pit, through which the air for combustion passes on its way to the kiln. This practice is similar to that frequently employed in portland cement burning. It has the advantage of recovering a part of the heat from the calcine and also any mercury vapor that is still escaping from the hot ore.

At Sulphur Bank an exceptional metallurgical problem was caused by the presence of elemental sulfur in the ore. In the early operations with the Scott furnace at Sulphur Bank, sulfur vapor distilled with the mercury. These two elements united in the condenser, and gave rise to large quantities of mercurial soot. The same difficulty was encountered when a rotary kiln was used with the customary countercurrent firing. Sulfur evaporates noticeably at a temperature below its ignition point and with

countercurrent firing it is inevitable that some sulfur vapor should be released in the upper part of the kiln where the oxygen content of the furnace atmosphere is low and the temperature insufficient for complete combustion. In the current operation at Sulphur Bank this difficulty has been overcome by reversing the method of firing; that is, by feeding the ore and firing at the same end of the kiln. With this practice the temperature throughout the lower half of the kiln is substantially uniform, thus providing a considerable zone in which the combustion of the sulfur vapor may be completed. With approximately 4 per cent oxygen in the furnace atmosphere and a temperature of 650° to 700° C. at the discharge end, no unburned sulfur escapes from the furnace. The sulfur dioxide formed shows no tendency to react with the mercury vapor and with the conditions just described no recombined mercury is found in the condenser product. With this practice the fuel consumption is necessarily increased because of the high temperature at which the furnace gases escape from the kiln; therefore provision must be made for extra cooling in the condenser system.

The Herreshoff Furnace

In recent years several Herreshoff furnaces of the familiar McDougall type have been installed for the roasting of quicksilver ores. The multiple-hearth furnace was developed for the roasting of sulfide ores, which ordinarily have a fuel value sufficient to make them nearly if not entirely self-roasting. The usual quicksilver ore has practically no fuel value; hence in applying this furnace to the quicksilver field it was necessary to devise a method for supplying heat from an external source. This problem has been successfully worked out.

Recently an 8-hearth, Herreshoff furnace, 15 ft. external diameter, has been installed at one of the Nevada mines, with two oil burners on both the fourth and sixth hearths. Ordinarily only the burners on the sixth hearth are used. Operating on −1-in. material, this furnace has handled 90 tons per day and shown a remarkably low fuel consumption of approximately 5 gal. of oil per ton of ore. This is 2 to 3 gal. per ton less than the best rotary performance of which the author is aware. This low fuel consumption is the natural result of the general design of the furnace; that is, a large hearth area contained within a relatively small volume as compared with the rotary.

It has not been shown that 90 tons per day is the maximum capacity of this furnace, and there is no apparent reason why the throughput can not be increased by at least 10 or 20 per cent.

The multiple-hearth furnace is essentially gas-tight and the ability to operate without gas leakage is important not only in avoiding loss of mercury but in guarding the health of the workmen. In contrast,

the upper seal ring of the rotary is usually more or less troublesome with respect to gas leakage.

As regards first cost, when comparison is made on the tonnage basis the installation cost of the Herreshoff compares favorably with that of the rotary. The performance shown by the Herreshoff furnace to date indicates that it has a real place in quicksilver metallurgy.

New Designs of Mechanical Furnaces

Several innovations in mechanical furnaces as applied to quicksilver metallurgy have been proposed in the last few years and some of them have been tried out in practice. Several of these designs have embodied the idea of a continuous muffle furnace; that is, a furnace in which the ore would be calcined in a closed chamber. The separation of the mercury-bearing gases from the fuel gases greatly simplifies the condenser problem. On the other hand, the indirect transfer of heat to ore through a separating wall is much less efficient than the direct transfer from a luminous flame and from the incandescent interior of the furnace. Furthermore, the rate of heat transfer is necessarily low. It follows that any apparatus working on the muffle principle will show a relatively high fuel consumption and a small capacity.

The designs include externally heated rotary shells and mechanically rabbled furnaces of iron and steel construction resembling the general design of the Hegeler furnace or consisting of a series of short sections of screw conveyor mounted in a brick chamber. The attempts that have been made along these lines are to be regarded as strictly experimental and have not thus far resulted in anything of practical importance. The standard types of furnace discussed above all give good metallurgical results as far as elimination of the mercury from the ore is concerned and are economical in their use of fuel. It is doubtful whether there is need for any essentially new design of furnace for treating the usual grade of quicksilver ores.

The Dust Problem

The dust problem was practically unknown in the quicksilver industry until the advent of the rotary kiln. The rotary, whether in quicksilver or other industrial uses, is generally recognized as a dust maker. The dust problem has been a troublesome one but in the last few years satisfactory progress towards its solution has been made.

Dust production varies widely with the character of the ore, ranging from 0.5 per cent. to as much as 5 per cent. of the furnace feed in extreme cases. The average is probably between 1 and 2 per cent. The condition of the lining of the rotary has a great part in dust production. Cracks and crevices in the lining serve as lifters, which shower the fine ore into

the gas stream. Even a small amount of dust in the gas stream entering the condenser system is highly undesirable. It tends to clog the condenser system and diminish its cooling efficiency. The troublesome job of cleaning the condenser system must be performed more frequently and the mixture of dust or mud with finely flowered mercury is a difficult product to treat for the recovery of the mercury.

In any method of dust collection the object is to remove as much dust as possible at high temperature. Dust so collected is a mixture of roasted and unroasted ore particles. The minimum temperature for collecting a low-grade dust is from 200° to 250° C. This is well above the mercury dew point for most furnace operations but a good margin of safety is necessary on account of local cooling along the walls of the dust-collecting device or through the inward leakage of cold air. If a panning test reveals metallic mercury in the dust it indicates that the temperature in the dust chamber is too low or that some local cooling is taking place.

The first attempts at dust control in connection with rotary kiln operations were through the use of dust chambers patterned after the familiar masonry chambers used in conjunction with the Scott furnace. These were relatively tall structures of small cross-section and were obviously not well designed from the standpoint of dust collection. A dust chamber is nothing more or less than a settling device. To be at all effective its horizontal dimension should be large in proportion to its height. A dust chamber of this latter type was originally installed at the Opalite plant of the Mercury Mining Syndicate. It consisted of two horizontal chambers each 16 ft. long, 5½ ft. wide and 6 ft. high. The walls were of concrete, 1 ft. thick, to provide good thermal insulation. Several vertical chain curtains were used to break up eddy currents and produce a uniform flow of gas. These two chambers operated in parallel, each handling approximately 1500 cu. ft. of gas per minute at a temperature of 200° C. to 250° C. The horizontal velocity of the gas stream through these chambers was less than 1 ft. per second, thus allowing over 16 sec. for the settling of dust particles. Theoretically, particles well below 200-mesh size should have come to rest in the dust chamber. Samples of dust actually collected showed that this was the case; nevertheless there remained in the dust stream a sufficient quantity of extremely fine material to give trouble in the condenser system.

The problem was solved by installing a Cottrell electrical precipitator. The kiln handles from 90 to 100 tons of ore per day and the Cottrell collects on the average some 1250 lb. of dust per day—a recovery of about 93 per cent. Approximately 1 lb. of dust per ton of furnace charge enters the condenser system, and this quantity has not been troublesome.

Water sprays have also been used for knocking down the dust. Mercury vapor is condensed at the same time and the product flowing

from the condenser system will be a mercury-bearing mud. In exceptional cases the mercury particles settle rather freely through this mud and a fair separation can be obtained by the use of riffled launders and settling boxes. Tanks with low-speed agitators have also been used. The separation just referred to seems to take place most readily when the dust partakes of the characteristics of clay and forms a colloidal suspension in the water. When the dust has the characteristics of an extremely fine sand rather than of a true slime it tends to settle with the mercury particles and no satisfactory separation can be made by mechanical means. The handling and retreatment of mercurical mud is disagreeable. Loss of mercury is bound to occur and a serious health hazard is involved. Frequently the mud is too low grade to justify retorting, and returning the mud or dust to the kiln tends to build up a heavy circulating load. Treatment of this material by flotation will be described later.

Electrical precipitation by the Cottrell process is undoubtedly the most satisfactory solution of this problem and several installations of the Cottrell process have been made in recent years. An alternative procedure, which has given fairly satisfactory results in a number of cases, is the use of cyclones of special design.

Cyclone Dust Collectors

Important pioneer work in the development of a suitable design of cyclone for this purpose was carried out at Sulphur Bank by the Western Precipitation Co. in cooperation with the Sulphur Bank Syndicate. The original design of the Sulphur Bank plant included a Cottrell precipitator or so-called hot treater. When the plant was put in operation it was found that the dust burden was much heavier than had been anticipated. The hot treater stopped a large amount of dust but the clearance was not as good as desired. The alternatives were to increase the hot treater capacity or to use some other auxiliary device. It was decided to investigate cyclones. Very little information on the theory of cyclone design was found in the literature. The cyclone is essentially a centrifugal device and it seemed clear that a design which provided for a high angular velocity coupled with a relatively short path for the travel of the dust particles should give the best results. This was confirmed by tests with several models. The work led to the design of tall, narrow cyclones with special provision for trapping the collected dust at the bottom of the cone. It was also found that by suitable design the draft loss through the cyclones could be reduced materially.

The cyclone installation at Sulphur Bank consists of 12 tall, narrow cyclones arranged in two sets of 6 each, so that either series or parallel arrangement can be used. Each set of cyclones is preceded by a so-called

"precharger," which consists of a small Cottrell treater of the pipe and wire type. In passing through this precharger the dust particles become electrically charged and pass in the charged condition into the cyclones. Agglomeration also undoubtedly takes place. The net effect of this precharging is a noticeable increase in cyclone efficiency. The cyclones are more effective when operated in series, owing to the higher velocity through the individual cyclones and the double treatment.

These cyclones are also remarkably efficient as coolers. As mentioned, the Sulphur Bank kiln is operated with parallel firing, so that the gases leave the kiln at high temperature. The cyclones operated in series cool the gas stream from an inlet temperature of 500° to 550° C. to an outlet temperature of 200° C.

High-velocity cyclones of generally similar design have also been developed by the Rees Blow Pipe Manufacturing Company of San Francisco. An installation of Rees cyclones is described by W. G. Adamson.[1] Where the dust problem is not too serious, high-velocity cyclones seem to offer a satisfactory solution. They are less expensive than the Cottrell equipment, but their efficiency cannot equal that of the Cottrell, and when a dust situation is serious electrical precipitation by the Cottrell process is undoubtedly the best procedure.

Condenser Practice

Masonry chambers of brick or stone were almost universally used in connection with early quicksilver furnace practice in this country. With the advent of the Scott furnace a condenser design consisting of a series of brick chambers became standarized, although in a few cases cast-iron chambers connected by large cast-iron U-bends were used. In 1917, the U. S. Bureau of Mines in cooperation with certain quicksilver producers conducted an investigation of quicksilver metallurgy[2] and in this connection an experimental condenser unit constructed of glazed sewer tile was tested. In Europe, condensers of chemical stoneware had been in use for some time. These tests were followed by the construction of a tile condenser at the Oat Hill mine, Napa County, California. Concurrently, condensers of glazed sewer tile and wooden tanks or chambers were constructed at several other plants. Since 1918, condenser practice has developed along the lines of what may be termed a pipe and tank construction. Steel or cast-iron pipe have been employed for the section of the condenser closer to the furnace in the temperature range

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[1] W. G. Adamson: Recent Progress in the Metallurgy of Quicksilver. *Eng. & Min. Jnl.* (1929) **128**, 504.

[2] L. H. Duschak and C. N. Schuette: The Metallurgy of Quicksilver. U. S. Bur. Mines *Bull.* 222 (1925) 129.

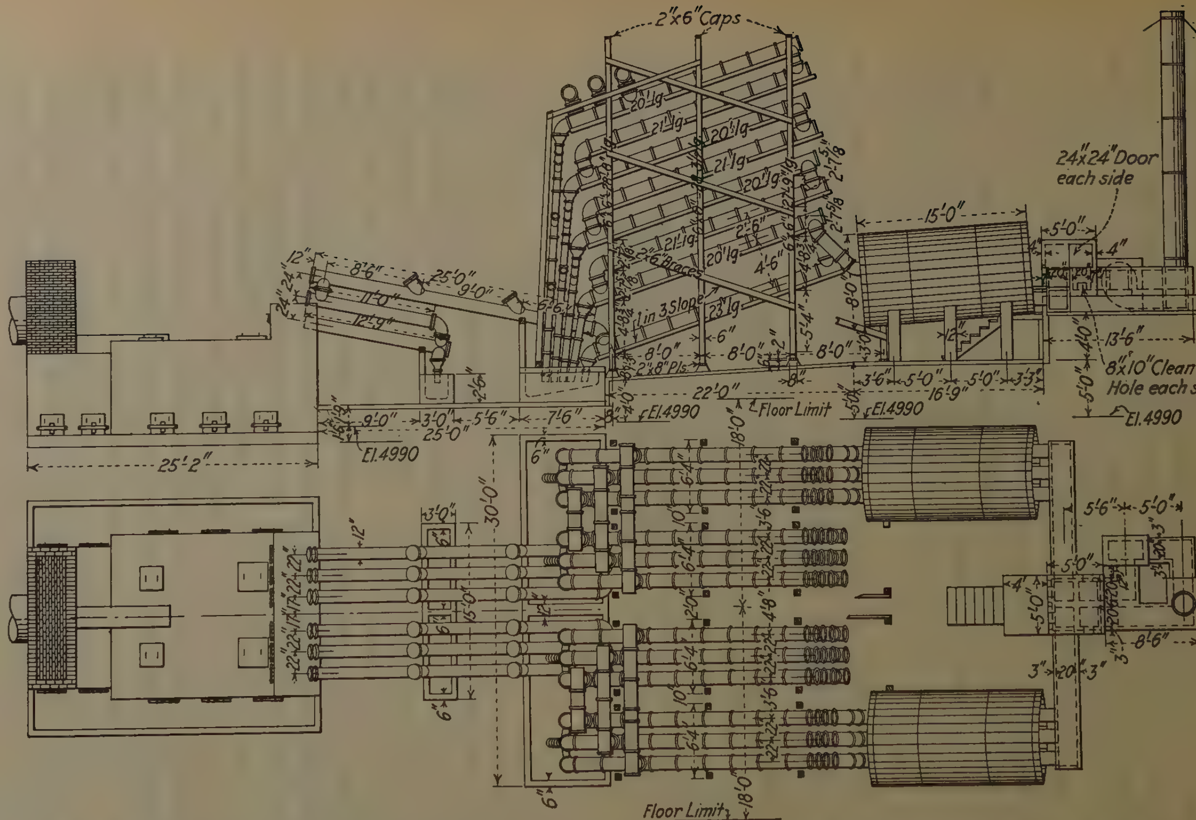

FIG. 3.—CONDENSER SYSTEM AT OPALITE PLANT OF MERCURY MINING SYNDICATE.

Horizontal arrangement of tile cooling pipes. Note that the tile pipes all drain into a common sump. Dampers are provided for cutting out either half of the condenser system so that one half can be cleaned while the other is in use without interrupting furnace operation. The Cottrell precipitator, which precedes the chamber at the left, is not shown in this drawing.

where little condensation of water occurs; glazed sewer tile and wooden tanks are commonly used for more remote parts of the condenser system.

Various arrangements have been used. These may be classified under three general heads; namely, (1) pipes slightly inclined from the horizontal; (2) pipes in the form of an inverted V with concrete connecting chambers at the base; (3) pipes in the form of an inverted U with concrete chambers at the base. The horizontal arrangement has the advantage that a large cooling surface can be constructed over a relatively small area, thus making for compact construction and minimizing the danger of mechanical loss of mercury. Pipes in the horizontal position give better opportunity for the circulation of cooling air or the application of water sprays, and therefore give a greater cooling effect per unit area. On the other hand, vertical pipes or pipes slightly inclined from the vertical, are more easily cleaned. The choice of design is largely a matter of preference and should be governed by local conditions.

There is evidence that there is a certain lag in the condensation of mercury vapor; that is, that an atmosphere supersaturated with mercury vapor may persist for an appreciable time in the condenser system. This is not surprising, in fact is to be expected, when one considers the extremely small concentration of mercury vapor in the condenser gases. For example, the gas stream leaving a furnace treating ore assaying 0.5 per cent. Hg will contain approximately 0.1 per cent. mercury vapor by volume. As the mercury vapor condenses, the concentration falls to an extremely small value and it seems inevitable that a certain time factor should be involved in the establishment of equilibrium. From a practical standpoint the existence of this lag points to the need for the inclusion of several large chambers in the condenser design. These chambers, which in practice take the form of wooden tanks, allow time for the supersaturation of mercury vapor to be relieved.

With a well-designed condenser system the loss of mercury from the stack is extremely small and is very little more than that corresponding to the normal saturation of the stack gases with mercury vapor. A furnace plant handling 100 tons of ore per day will discharge from 3000 to 5000 cu. ft. of stack gas per minute. This corresponds to 6 to 10 tons of gas per hour. The mercury contained in this weight of gas may range from 0.5 to 1 lb. Viewed in this way, the performance of quicksilver condensers is remarkable.

Water Sprays Seldom Desirable

In a number of plants water sprays have been used for the direct cooling of the mercury-laden gases. It should be noted that the spraying of a stream of hot gases with water actually removes very little heat from the gas stream but merely converts sensible heat into the latent heat of water vapor. The only heat actually removed from the gas

stream by the water is that carried away by the portion of the water that does not evaporate. A simple calculation will show that the actual removal of any large quantity of heat in this way calls for the use of a volume of water which is entirely impractical. The impression that water sprays give effective cooling arises naturally from the drop in temperature that takes place and it is not so apparent that what has actually happened is not a removal of heat but a conversion of sensible heat into latent heat.

There is a fundamental objection to the use of water sprays; namely, the production of finely flowered mercury. As a general principle, the introduction of water to the condenser system should be avoided. With dry condensation, as the temperature of the gas stream is reduced the saturation point for mercury vapor, or what may be termed the dew point, is reached well before the gas stream becomes saturated with water vapor. Under these conditions a large part of the mercury vapor condenses directly on the cooling surfaces. Small droplets of mercury are formed which gradually grow by the direct condensation of mercury vapor. Moreover, the large surface of liquid mercury exposed in this way favors the release of supersaturation. When a water spray is used the minute mercury droplets produced by the initial cooling effect of the spray become filmed with water and are thus prevented from coalescing with one another.

When the dust problem is particularly serious, and when in spite of dust-collecting devices a considerable amount of dust passes through to the condenser system, water sprays may be advantageous in knocking down this dust and preventing it from fouling the cooling surfaces. When this practice is followed suitable provision must be made for recovering the mercury from the muddy water leaving the condenser system.

Treatment of Condenser Products

Reference has been made to the use of settling boxes and slow-speed agitators for separating finely divided mercury from condenser mud. Both horizontal tanks fitted with a horizontal shaft and paddles in some ways resembling a log washer, and circular tanks with slow-speed paddle agitators, have been used. Devices of this sort will recover a certain proportion of the mercury in fairly concentrated form but the overflowing mud is usually too high grade to be discarded. When there is any quantity of mud to be treated, flotation is undoubtedly the best method. At Sulphur Bank a two-cell Kraut unit is used for treating the mud. A mud carrying 5 per cent. mercury will yield a 50 to 70 per cent. Hg concentrate and a tailing carrying only a few pounds per ton dry weight. Tests have demonstrated that mercury sulfide formed

by the recombination of mercury and sulfur in the condenser is as readily floatable as finely divided metallic mercury.

For the final recovery of the mercury from high-grade condenser products or from a flotation concentrate, D retorts still remain standard. At Sulphur Bank an electrically heated drying pan, furnished by the Joshua Hendy Iron Works, has been found satisfactory for working over high-grade material prior to retorting.

Hydrometallurgical Processes

It is well known that mercury sulfide is soluble in a dilute solution of sodium sulfide. In 1916, Thornhill[3] described a method for the recovery of mercury from the amalgamation tailing at the Buffalo mines, Cobalt. This was based on the use of sodium sulfide solution as a solvent and metallic aluminum as the precipitant. From time to time attempts have been made to apply this procedure to the treatment of ores, electrolytic precipitation being proposed in some cases in lieu of precipitation by metallic aluminum. To date no practical application of this process has been forthcoming.

Tests that were made under the supervision of the author showed that one of the fundamental difficulties is the excessive destruction of the solvent. Sodium sulfide in dilute solution is extremely sensitive to oxidation by atmospheric air and this oxidation is catalyzed in an extraordinary way by the presence of iron oxide. Sodium sulfide in solution reacts readily with limonite or hematite to form ferrous sulfide and the films of ferrous sulfide so formed are rapidly oxidized by dissolved oxygen. Moreover, the reaction between ferric oxide and sodium sulfide results in the destruction of a certain amount of the sodium sulfide, since the iron is reduced to the ferrous condition. As practically all quicksilver ores contain more or less oxidized iron, the difficulty is fundamental.

Several processes based on the use of chlorine or hypochlorite in aqueous solution have been proposed recently. These reagents will act as solvents for mercuric sulfide but no practical means have been developed for the recovery of mercury from the resulting solution. There are also other drawbacks.

In this connection, it is well to point out again that the direct furnace treatment of quicksilver ores is not only simple and inexpensive but metallurgically efficient. The quicksilver industry presents no general complex ore problem. In these circumstances there is no apparent place for hydrometallurgical methods in the quicksilver industry.

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[3] E. B. Thornhill: Wet Method of Mercury Extraction: *Min. & Sci. Pr.* (1915) 110, 73.

Summary

The direct furnace treatment of quicksilver ores remains the standard practice and when suitable attention is given to the design of the plant leaves little to be desired. The problem of treating a large tonnage of ore below the necessary furnace grade may be solved under favorable conditions by wet screening followed by flotation and the retorting of the concentrate. There is a very apparent need for economical methods for use during the development of a property or where operations are necessarily on a small scale. Hand sorting and retorting have been the chief reliance in this connection. However, this practice is successful only where a relatively high-grade product, say 5 to 10 per cent. Hg, can be economically produced by sorting. When this is not possible a small furnace plant or a small flotation plant and battery of retorts offer the best solution.

[For discussion of this paper, see page 312.]

The Present Status of Our Quicksilver Industry

By Charles G. Maier,* Berkeley, Calif., and Contributors

(San Francisco Meeting, October, 1929)

In preparing this symposium, our ambition was to elicit authoritative expression of opinion concerning important selected phases of the industry from men active in it. Responses to requests for contributions were gratifying, especially so in view of the short time available. The generous and cooperative spirit of the industry as a whole is amply evident. Responses were received from the following, who are therefore the true authors of the paper:

W. G. Adamson, General Manager of the Pershing Quicksilver Co., Lovelock, Nev.; Dudley Baird, Vice President, Pacific Foundry Co., San Francisco; Walter W. Bradley, State Mineralogist of California; W. D. Burcham, General Manager of the Brewster Quicksilver Consolidated, Terlingua, Tex.; William Forstner, San Francisco; H. W. Gould, of H. W. Gould & Co., San Francisco; Marcus Hulings, General Manager of the Chisos Mining Co., Terlingua, Tex.; Frank J. Katz, Chief Engineer, Division Mineral Statistics, United States Bureau of Mines, Washington, D. C.; Lloyd J. Lathrap, Superintendent, Nevada Quicksilver Mines, Inc., Lovelock, Nev.; W. R. Moorehead, General Manager, New Idria Quicksilver Mines, Inc., Idria, Calif.; F. W. Oakes, Jr., Terlingua, Tex.; C. N. Schuette, San Francisco.

Brief Economic History of Quicksilver in the United States

Quicksilver ore was first discovered in the United States at what was later the New Almaden mine in California, in the year 1824. Production also began here, but not until 1850, when it was stimulated by the active demand caused by gold-mining operations. Prospecting for quicksilver was active, high-grade ores were found and ample production lowered the price from $100 per flask in 1850 to about half that figure for the greater part of the next 10 years. During the succeeding decade, many new quicksilver mines were found; production increased at a time when the local demand diminished, so that most of the product was exported. The third decade of quicksilver mining in the United States witnessed the development of the pan amalgamation process, particularly in Comstock silver-mining operations. A suddenly increased demand sent prices soaring, the all-time peak of quicksilver production followed, and the highest price was soon replaced by the lowest one on record. Huge quantities of quicksilver were exported between 1870 and 1880 and exports continued in decreasing measure for three decades to 1910. In

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the years before the world war the United States produced just enough to cover consumption requirements. High prices during the war stimulated production and depleted the developed and the readily available reserves of low-grade dump material. After the war, Forstner says, "the price declined as expected. When the prices dropped from $117.50 per flask in 1918 to $45.50 in 1921, a number of operators, more than 60 per cent. of those operating in 1919, closed down. When prices gradually rose, some started up again, but this movement did not get its main impetus until it became known that the Italian producers and the Spanish government, which controls the Spanish quicksilver deposits, were negotiating for an agreement to cooperate in the production and marketing of quicksilver, with the view of sustaining the market price, which agreement was finally concluded in July, 1928." At present production in the United States is stimulated by high prices and has climbed back from a low of 6339 flasks in 1921 to the respectable total of 18,108 flasks in 1928. According to Schuette (in a forthcoming publication on quicksilver of the Bureau of Mines) consumption in 1928 was around 35,000 flasks and it will take domestic producers a few years more to supply such a demand from domestic sources. We have had of late years an increasing consumption during a time of increasing prices. This augurs well for the future and seems to prove the indispensability of quicksilver in industry.

In this connection, the Pacific Experiment Station of the U. S. Bureau of Mines undertook during the past year to determine as far as possible the actual present modes of consumption. Through the generous cooperation of consumers of quicksilver, it was possible to account for more than 90 per cent. of the quicksilver used in this country. Details of these results are to appear in the above mentioned bulletin; but certain general facts deserve reporting at this time. Scientific, technical control, and electrical apparatus consume some 20 per cent. of the total, and such dwindling amounts as for amalgamation are more than made up by these relatively new uses. The drug and chemical trade absorbs some 30 per cent., the manufacture of fulminate 20 per cent., felt manufacture, pigments, and various miscellaneous uses consume the remainder. This superficial summary gives no true idea of the great diversity of uses. Diversity of use means a tendency towards price stability. The high price brings lower grade material into the category of ore and several United States mines are profitably treating ore that contains an average of only 3 lb. of quicksilver per ton.

Geology and Mining of Quicksilver

The U. S. Bureau of Mines[4] in 1925 pointed out that the greatest opportunity for increasing economy in quicksilver production lay in

[4] L. H. Duschak and C. N. Schuette: Metallurgy of Quicksilver. U. S. Bur. Mines *Bull.* 222 (1925).

giving more attention to the geology of the deposits and the improvement of mining methods. Schuette has postulated a theory of the occurrence of quicksilver ores which points out that quicksilver orebodies as differentiated from uncommercial occurrences are due to a concentration of the primary mineralization during deposition. This theory was developed from a study of quicksilver orebodies the world over and the largest and richest orebodies most nearly fulfilled the precepts of the theory. It is further pointed out that quicksilver orebodies as formed by primary concentration need not outcrop, that they have distinct characteristics that aid in finding them and they often form high-grade orebodies. The finding of most of the present day quicksilver mines was due to fortuitous discovery rather than the result of intelligently directed prospecting and it may be postulated that relatively more deposits of this type remain to be discovered than of types that outcrop prominently.

One of the most interesting development programs is that of the New Idria Mine, as told by Mr. Moorehead. The orebody strikes east and west. It outcrops on the north slope of a mountain for some 1000 ft. along the strike. The deposit dips south. This orebody was developed and mined by tunnels to a depth of 1000 ft. below the outcrop and thus for 400 ft. lower by a shaft from No. 10 tunnel. Underground prospecting is largely done by a prospecting or deep-hole drill. Holes up to 275 ft. have been drilled in this work, using 1¼ in. steel. For best results the holes are pointed up at a slight angle and a number of holes are fanned out from any one set up. Care is needed in drill operation to insure a correct interpretation of results when ore is found. The amount of water used in drilling must be controlled within certain limits and it is best to proceed for only short distances, after which the hole must be cleaned. The ore content is estimated by panning the sludge. The operating cost for such drilling, including all charges and the cost of all equipment for a campaign of 20,000 ft. of drilling will not exceed 90 c. per foot.

A comprehensive scheme of development for the low-grade ore in the old upper workings of this mine is now under way. It is intended to work the entire tonnage of the ore zone from the outcrop to the 300 level by open-pit or quarry methods with shovels and trucks. To do this it is necessary to strip the overburden from the hanging wall side of the ore zone. This is being done by the glory hole method through a raise run up from the 300 level through the hanging wall. The hanging wall is milled into this raise, drops down to the 300 level and is carried out through it on a 43-in. conveyor belt 1200 ft. long. The tunnel level is 750 ft. long and the belt runs 450 ft. out on the waste pile where the waste is dumped by a distributing tripper.

It is estimated that about 2,000,000 tons of overburden will have to be removed in order to recover the entire reserves from the surface to

the 300 level. At the present time approximately 1,000,000 tons of this overburden have been removed.

The tonnage mined from Idria quarry will be sent down an ore chute to the 500 level. Ore mined between the 300 and 500 levels by square set or shrinkage system will also be transported through this chute.

In order to reopen the old levels and sublevels from the 500 to the 1000 level, which is the main haulage level, and recover the available tonnage of ore in this area, a two-compartment incline raise is being put up in the footwall connecting the 1000 and 500-ft. levels. A hoist will be installed on the latter with two skips in counterbalance having a lowering capacity of 5 tons each. This installation will not only serve to open up all of the old levels and sublevels but will provide a cheap method of transporting the ore from the 500 to the 1000 level and thence the ore will go by battery locomotive through No. 10 tunnel to a screening plant. The 500 tunnel haulage system and the tramway from it to the plant will be abandoned.

It is expected that the quarrying operations from the outcrop to the 300 level will make available 750,000 to 1,000,000 tons of rock that will run between 2½ and 3 lb. of quicksilver per ton.

This will all be treated in a picking and screening plant now under construction, in which it is expected to realize about a 4 to 1 concentration ratio with an 80 per cent. recovery. Thus the ore development project should yield about 250,000 tons of about 8-lb. ore.

The Last Scott Furnace?

The greatest part of the quicksilver produced in the United States has come from Scott furnace plants, but since the war mechanical furnaces have been supplanting it, though it still accounts for most of the Texas productions.

W. D. Burcham built the latest Scott furnace to be erected and furnished the following interesting data concerning it:

"This furnace was built at the Mariscal mine some 40 miles from the Terlingua district in Texas. It is of 50-ton capacity rating, that is, four tile and four shaft, and was built on the north bank of the Rio Grande, 100 miles from railroad transportation.

"This furnace differed from the usual type in only one particular. The customary timber buckstays were replaced with structural steel, the uprights being six inch channels and the belts being eight inch I-beam with 1⅛-in. truss rods. These held together at the corners with 1⅛-in. bolts under the nuts and heads of which were 50-ton car springs. The object of this arrangement was to keep the tension constant. It was apparently satisfactory as no cracks of any magnitude developed in the brick work. It was built by day labor at the following costs: The skilled labor cost $11 per day for the foreman, $9 per day each for four brick

layers and $7 for the mortar man from the time they left San Jose, Calif., until their return, together with all expenses to and from the job and board and lodging while on the job. Common labor cost $1.25 per day. Two hod carriers were provided for each brick layer and several roustabouts were kept on hand, all of whom were under a foreman. The time of construction was 52 days. Total cost of the furnace is shown in Table 1.

"The cost of this furnace, which was not housed and which does not include the stone work condensing system but which was built far from transportation and under difficulties that only those familiar with this isolated location can appreciate, was then something less than $400 per ton-day capacity."

TABLE 1.—*Total Cost of Scott Furnace*

Lime, 350 bbl. made on the ground at $1.20		$ 420.00
Common brick, 160,000 made on the ground at $20 per M		3,200.00
Firebrick, 21,400	at $93 per M f.o.b. cars Marathon, Tex.	1990.20
Arch firebrick, 1,000		93.00
End skew firebrick, 7,800		725.40
Side skew firebrick, 1,700		158.10
3 × 15 × 36 in. fire tile, 424 at $3.75		1,568.80
Fireclay, 45 bbl. at $4		180.00
Cement, 60 bbl. at $4.40 per M f.o.b. cars, Marathon, Tex.		264.00
Sand, 200 tons		200.00
Structural steel for buckstays		1,724.56
Iron work, as per Scott's drawings		993.44
Lumber		184.00
Total cost of materials (including railroad freight)		$10,901.50
Skilled labor		$ 4,628.00
Unskilled labor		1,170.00
Wagon freight, railroad to mine, 311,579 lb. at 75 c. per cwt.		2,336.84
Mrs. Scott, for plans		500.00
Total cost of labor and plans		$ 8,634.84
Total cost of furnace without housing		$19,536.34

Marcus Hulings of the Chisos mine, where a Scott furnace has been in operation since the mine started and where a rotary kiln furnace was operated for a number of years, contributes the following:

"I am not qualified to make an authoritative, detailed comparison between the Scott and the rotary furnaces, but a comparison in a general way might be of interest. It can be assumed, with some degree of accuracy, that either type properly built and properly handled will extract practically all the mercury from the ore charged into it. In the late practice of condensing, pertaining to the rotary, the mercury is recovered almost simultaneously with condensation. What percentage the recovery may bear to the mercury contained in the ore will depend on the degree of perfection with which the plant is manipulated; very

little quicksilver is held in suspension within the condensers. With the Scott furnace and the old type of brick condensers, great quantities of quicksilver adhere to the walls of the condensers and gradually penetrate the mortar joints, in instances almost to saturation, and even the bricks themselves are permeable to a certain extent. This is the outstanding undesirable feature of the plant. Other types of condensers than brick, used in conjunction with the Scott furnace, partially obviate this contingency.

"The flexibility of the rotary, in lending itself readily to meet demands of desire or necessity, is an advantage. The heat may be controlled by the turning of a valve. The tonnage rate may be governed by regulating the speed at which it is being run. It may be stopped or started at will, with no resulting damage. On the contrary, the Scott permits of little deviation from one prescribed course. It can only be stopped, cleaned out and started, at the expense of considerable time and trouble.

"The rotary, with its condensing complement, comprises a compact and simple installation, mostly housed under one roof. The equipment is all fabricated, can be purchased in the market and erected at a cost of time and money much less than that required to install a Scott plant. The Scott with its condensing apparatus is an extended array of units, consisting of the furnace itself and the various condensers.

"Progress advances with the times. New methods of reducing quicksilver ores supersede the old. The Scott furnace has had its day, and must, more or less gracefully, recognize that it is becoming ancient and give way to the younger element. But the Scott furnace in its surrender, might truthfully avow that it was strong in its youth, far surpassing all competitors and that, even in its old age, it may still be, due to conditions of isolation or water scarcity, more feasible than some later invention.

"It is true that a well-seasoned, saturated Scott furnace functions beautifully. It is practically automatic in its operation. Little attention is required to keep it in proper working order. The percentage of recovery is of the highest. There are no mechanical breakdowns, no perforated condensers, nothing to disturb assured, continuous uninterrupted performance of duty."

The Day of the Rotary

According to H. W. Gould, the rotary kiln was first successfully applied at the New Idria mine in 1917. As first developed it was thought to be suitable for low-grade ores that made relatively little dust in treatment. In the early installations dust was separated from the hot gases in dust chambers and by means of sprays in the pipes leading to the condensers. In 1926 at the Knoxville plant a heat-insulated cyclone dust collector was tried between the furnace and the condenser. The instant success

achieved has resulted in the wide application of this device for both rotary and Hereshoff furnaces.

The cost of rotary kiln plants varies with the location. Simple plants with a 4 by 60-ft. kiln treating 80 tons per day can be built for about $32,000 or $400 per ton-day capacity. Smaller kiln plants, say with 3 by 40-ft. kilns, will cost from $500 up per ton-day capacity. In a general way, the cost varies between $400 and $1000 for ton-day capacity, depending on the type of kiln, power plant, condenser, buildings, crusher and bins used.

The suitability of properly designed rotary kiln plants for the treatment of high-grade ore has been demonstrated at the Nevada Quicksilver Mine. Lloyd J. Lathrap contributes the following:

"The ore is a brecciated limestone, and as fed to the furnace has varied from 8 to 80 lb. of quicksilver per ton. The general average is in the neighborhood of 30 lb. The ore coming from the mine is passed over a grizzly with bars spaced at 2-½ in., the undersize going direct to the furnace bins. The oversize is passed to a crusher with a 2-in. opening and then to the furnace bins. A considerable portion of the ore, however, breaks fine and at times the furnace feed is largely material under 1 in. dia. and in part from ¼ in. down to dust. The plant, which has been described in detail by H. W. Gould,[5] consists of 3 by 40 ft. rotary furnace, equipped with cylcone dust collectors, and a series of eight pairs of monel metal condenser pipes 18 in. dia. by 18 ft. high, followed by three 10 by 20 redwood settling tanks and a 30-in. redwood stack 30 ft. high.

"The furnace is set at a slope of ¾ in. per foot and at the start of operations was turned 1 r.p.m. This has since been increased to one revolution in 40 sec. and at this speed the ore passes through the furnace in 30 min. The amount of feed is varied according to the character of the ore; the daily capacity of the furnace ranging from 30 tons with fine dusty ore up to 45 tons with clean coarse ore, with the average a little better than 35 tons for operations to date.

"Because of the presence in the ore of numerous coarse pieces of practically pure cinnabar, some up to largest size of furnace feed, it has been found necessary to carry a higher temperature than is usual in rotary furnace operation.

"The volatilization of the cinnabar in such pieces depends not so much on time in the furnace as on temperature. When the rock discharges from the furnace at temperature under a bright cherry red, the extraction from these heavy cinnabar pieces is not complete. The higher temperature is necessary only for the coarse pieces of solid cinnabar. Even high-grade ore, where the cinnabar occurs as fairly small particles, well

[5] Gould, H. W.: Nevada Quicksilver Mines, Inc., Operates New Plant. *Eng. & Min. Jnl.* (1929) **127**, 9.

disseminated through the rock, will be completely burned when emerging from the furnace at a dull red heat.

"The most notable point in connection with operations here is the early recovery of quicksilver. Of the total recovery 90 per cent. is secured from the first three pairs of pipes and beyond the fourth pair the collected soot is not of sufficient value to pay for any treatment other than drying and refeeding to the furnace. This early recovery is certainly due in part to the quick cooling in the thin metal condensers, but it is also largely influenced by the grade of the ore and the resultant fumes. Apparently the condensation from the high-grade fumes is in the form of coarse particles which settle more readily from the gas stream. On ordinary days, with a 10 or 12-flask clean-up, the recovery has been approximately 20 per cent. from the first pair of pipes, 50 per cent. from the second, 20 per cent. from the third, and 10 per cent. from the fourth pair. On days with a 30-flask clean-up (and there have been a number such) approximately 50 per cent. has come from the first pair of pipes, 30 per cent. from the second, 15 per cent. from the third, and 5 per cent. from the fourth pair.

"The phenomenon is noticeable throughout the system, even to the stack. While the stack loss (as evidenced in gold foil tests) has at no time been large, there have been times when the observed loss was less with high-grade ores than with ores of medium grade.

"Operations here have demonstrated clearly the possibility of treating in a rotary furnace ores of high grade, with a most satisfactory overall recovery."

Monel Condensers

The use of monel metal condensers originated at the plant of F. W. Oakes, Jr., in the Terlingua district of Texas, and he contributes the following comment concerning their use:

"The writer had long been of the opinion that monel metal, or particularly Duriron, would make excellent condensing systems for quicksilver plants. Monel metal was chosen because of its lighter weight and its acid-resisting qualities.

"The condensing system is of welded 18-gage monel metal manufactured by Rees Blow Pipe Mfg. Co. It consists of two cyclone dust collectors, insulated by 4 in. of Diatomite furnished by Denver Fire Clay Co., thence the fumes pass into a 20-in. monel-metal exit pipe which in turn leads down into a 30-ft. junction box in five sections. From the top of each section return pipes 20 in. by 20 ft. lead into the next section, and to the end of this system another larger cyclone is attached.

"From here an exit pipe leads into a series of three black iron tanks, similar to the typical California redwood barrel-form tanks, and thence to the stack. The stack was originally designed to be on top of the

cyclone at the end of the monel metal system, but considerable quicksilver was carried with the dust to this point and the outside tanks were therefore installed.

"Although the first cyclone must be cleaned every day, the second cyclone collects very little. In two weeks continuous run less than a wheelbarrow load accumulates here, but is fairly rich, and is put into the furnace. The first cyclone apparently does its work well, and the second one is perhaps not needed although it acts as a valuable factor of safety.

"A Brown recording thermometer is placed in the first exit pipe immediately beyond the second cyclone. The temperature at this point is kept between 380° and 400° for best results. No effort has been made to take the temperature in the first cyclone, inasmuch as it functions perfectly, and is necessarily at much higher temperature than the exit pipe.

"Most of the quicksilver is deposited in the second and third sections of the monel system. About 60 per cent. of the metal is recovered as free metal or worked out on a soot table. The remainder is recovered in a retort. Moisture is regulated by a damper on the stack and its condensation is kept at the center of the system for best results. All of the material deposited as dust or mud beyond the third section is evenly distributed, and grows less until the last tank and stack is reached, where very little accumulates. No appreciable difference is noticed in the cyclone at the end of the system, hence the writer is firmly convinced that this cyclone is unnecessary.

"The outside tanks, made of black iron, are attacked by the sulfuric acid, but are easily patched or replaced. It has not been necessary to replace one as yet though several small holes have appeared. Wooden tanks might seem preferable, but in this hot dry climate they would be useless because of shrinkage.

"When the furnace was first placed in operation, brick from an old Scott furnace was run through it. Within 10 days it was noticed that the monel metal was definitely affected. Copper stains were seen at places, and a small portion of the system had to be replaced. Chlorides were found in the brick, and they formed a dilute hydrochloric acid, which immediately started to corrode the monel metal. Samples of the metal showed that it was very brittle and had lost much of its life and strength. Analysis of the metal showed: Chloride, 0.03 per cent.; mercury, 0.26; sulfur, trace. The furnacing of the brick was stopped immediately and the systems were repaired. Since that time cinnabar ore only has been treated. The furnace has been in operation one year, and there is no sign that the sulfuric acid has affected it in any way, but *it is highly important to note that monel metal condensers will be a failure where chlorides are present in the material fed to the furnace.*

"A certain amount of unburned cinnabar is found in the dust chamber at the feeder end of the revolving tube. The ore is so dusty that the exhaust fan picks up some values with the dust before the fire is reached. The extraction from the furnace has nevertheless been very high, some 96 per cent., based on assays of ore. It is evident that the loss is low and the monel system generally successful.

"Because of the increased radiation, the ease of erection, the very small 'hang up,' and the low weight to be taken into consideration when long hauls are necessary, the writer feels the monel metal system is entirely adequate provided chlorides are not present in the material to be fed into the furnace. The cost of such metal as compared to tile pipe, etc. is high, and unless hauling charges equalize this he does not feel that the advantages of monel metal justify the additional expense for such an installation if other materials are available and water is available to be used with sprays. He heartily recommends two cyclones at the head of any system, but does not consider a cyclone at the end of the system necessary. He must add the precaution that rivets should not be used in the monel condensing system. Expansion and contraction tear the metal away and necessitate patching."

The Hearth Furnace in Quicksilver Metallurgy

The use of the Herreshoff type of roasting furnace for treating quicksilver ores has had an interesting history, which would require too much space to describe fully. Dudley Baird has furnished the following details: "It was first applied to the quicksilver industry in 1916 at the Senctor Nina, New Almaden. Furnaces approximately 16 ft. outside diameter, with six hearths, were first built, to burn fuel oil or wood fuel. These furnaces calcined from 40 to 50 tons of ore per day, which has been crushed to about 1 in. Firing was done at the fifth hearth, the one next to the bottom.

"Herreshoff-type furnaces are manufactured in sizes ranging from 10 ft. 10 in. up to and including 21 ft. 6 in. in diameter, with from 6 to 16 hearths and therefore adaptable to almost any size plant. The latest and most improved installation of Herreshoff furnace is at the Pershing Quicksilver Mine, near Lovelock, Nevada."

W. G. Adamson writes: "What is believed to be the only wood-fired multiple-hearth furnace ever used in the extraction of mercury from its ores was erected and operated on the Goldbanks property, 38 miles south of Winnemucca, Nev., in 1914.

"It was of the standard type six-hearth (exclusive of the top, or drying hearth) 18-ft. dia. Herreshoff roasting furnace. The fuel was 4 ft. juniper cord wood, burned in two external fire boxes, constructed of brick and heavily insulated. The hot gases from one box were conducted through a flue to the third hearth from the bottom while the gases from the other box were conducted to the next hearth above.

"The furnace treated from 40 to 60 tons every 24 hr., depending upon the character of the material being put through and the degree of fineness to which it was crushed. It produced in all over 3000 flasks of quicksilver. The time required for the ore to pass through the furnace varied from 1 hr. 20 min. to 2 hr. 40 min. The fine crusher was set to ½ in., but at times may have had a maximum opening of ¾ inch. The ore was hard silicious or opaline and required a longer calcination time than is ordinarily considered necessary.

"The sixth or top roasting hearth was automatically sealed by ore passing through the peripheral openings from the drying hearth, and the exhaust gases containing the mercury vapor were drawn from the top roasting hearth at a temperature of approximately 500° F. into one of the old-type baffle dust collectors of about 12 by 12 by 18 ft. dimensions. From there the gases were conducted into 36 cast-iron pipes, 6 in. by 16 ft., which were jacketed with sheet metal. A Buffalo blower served to circulate cooling air about them. A fan placed between the iron condensing pipes and a tile pipe, 22 in. by 30 ft., pulled the gases from the furnace through the dust baffles and iron condensing pipes into the tile pipe and thence into four 10,000-gal. redwood settling tanks. Over 70 per cent. of the quicksilver was condensed in the iron pipes, which were topped with a ground joint iron cap and brushed down or cleaned with a dry swab daily. Due to the position of the fan, the draft was inward at all points in this unit, which minimized the loss of quicksilver vapor leaks in the system. No doubt a portion of the success met with in the operation of this furnace treating a difficult ore might be attributed to the use of wood as a fuel. We believe that a more complete combustion and a cleaner fuel gas with a higher reducing power can be obtained from the wood than from either oil or coal. The wood consumption varied from 1½ to 1¾ cords for every 50 tons treated. This also practically represents the daily consumption of fuel. The wood was a native juniper (locally known as cedar) and was cut and hauled to the mill on contract by Indians for $7.00 a cord.

"One feature of the rabble-stirred furnace that was particularly adaptable and meritorious at Goldbanks was the freedom from lining abrasion. The average thickness of the layers of ore on the various hearths was between 2 and 3 in. The teeth of the rabble have a special setting which provides for a practically stationary thin layer of ore next to the brick linings. These linings showed but little wear during the period of operation whereas some rotary kilns are troublesome in this respect, and may require relining several times a year.

"As is well known, the multiple-hearth furnace was designed primarily to roast fine ore. Due to the resistance to vaporization of the cinnabar in the Goldbanks ore, this finer crushing resulted in a shorter roasting period than otherwise would have been necessary.

"As is also well known, the hearth-type furnace was designed primarily for the roasting of material which contained a portion or all of its own fuel, and in the operation of the furnace with external fuel it was demonstrated that a much greater control of temperatures could be obtained on the hearths where it was advantageous to do so and with a resulting saving of the total amount of fuel consumed. The main problem on the other hand is to secure complete combustion in whatever fuel is used, and this can be done admirably in the case of wood, and quite successfully with oil.

"Moisture content of the ore has a considerable effect on the items of capacity and fuel consumption in the rotary furnace. An appreciable amount of wet ore will seriously lower capacity and raise fuel consumption. The top or drying hearth in the multiple-hearth furnace seems to be quite efficient in reducing the moisture hearth, and no appreciable difference can be noted in capacity or fuel consumption."

The Pershing Plant has a 4 by 60 Gould rotary furnace and 8-hearth 16-ft. dia. Herreshoff furnace. Both are equipped with two cyclones to remove dust from the gases and have similar condenser systems.

The ore coming to the furnace plant is crushed to 1½ in. in a 10 by 16 Blake-type crusher. It is then elevated to a ½-in. vibrating screen, the oversize going to the rotary-kiln bin and the undersize to the Herreshoff furnace.

The rotary kiln revolves at a rate of 72 revolutions per hour and the ore passes through the kiln in 55 minutes, being burned to a red heat.

The time of residence of the ore in the Herreshoff furnace is almost twice as long. The rotary kiln treats about 50 tons per 24 hr. and the Herreshoff about 85 tons, the latter treating all of the fines in the ore, as only some 35 tons per day were handled by the kiln before the Herreshoff was installed.

As now operated the kiln burns 7.2 gal. of 27° Bé. oil per ton of ore while the Herreshoff uses only 5.4 gal. per ton.

The Industry and Governmental Agencies

There is a final subject of interest in this symposium: the relation of quicksilver industry to various government agencies. The U. S. Bureau of Mines has attempted to aid the industry through its Pacific Experiment Station, by cooperative work in determining losses and by publications on various phases of the work. The U. S. Geological Survey furnishes topographical maps and geological data on the quicksilver occurrences. Large areas in the quicksilver districts of the United States remain unsurveyed, not even topographic maps being available, to say nothing of geological maps. It behooves the quicksilver industry to make a special plea for action in this matter, as accurate geologic maps are of utmost importance to the search for new orebodies.

With reference to mineral statistics Frank J. Katz writes: "The U. S. Bureau of Mines sends to every known producer of quicksilver an annual inquiry schedule. The form is simple and designed to develop information as to the amount of ore treated, quicksilver recovered, and the quantity and value of quicksilver sold. By furnishing to the Bureau of Mines a report on his own production, each operator enables the Bureau to furnish him with an authoritative statement as to the total output, subdivided, so far as practicable, to show the output in each of the important producing areas. The statistics of the complete 'Mineral Resources' are the results of simple summation from the reports of producers. Regarding the apparent lag of time between assembling statistical data and its appearance in official publications Mr. Katz says:

"The time lag on statistics is much less than commonly believed, for it is usually only a matter of a few weeks, generally less than a month, between the time of receipt of the last needed report and the appearance of our mimeographed release giving the essential summary of statistics. We issue press releases as soon as tabulations are complete, in order that information shall not be withheld unduly. Of course the matter of difficulty and delays in the writing of the final reports, their editing and their printing, is one concerning which some discussion might develop helpful suggestions. I would suggest also consideration of methods of inducement through which the mineral producers will respond promptly when they receive our inquiry schedules."

The California agency, according to Walter W. Bradley, endeavors to keep in touch with quicksilver producers by visits of field men, and by information relating to operations gained from the press. A simple form requesting information on the year's operation is mailed to each known producer during the last week in January of the following year. Bradley says: "One slight element of confusion has been the shifting from the old Spanish flask to 76¼ lb. to 75 lb., and now again to 76 lb. It is important that tonnages of ore treated should be recorded, as by those data ore grade percentages can be shown.

"California was for many years practically the only domestic producer of quicksilver, and still leads, though Texas and Nevada (followed by Arizona, Oregon, Washington, Idaho, Alaska in lesser amounts) are now contributing important quantities. For that reason, and the fact that large amounts of quicksilver were used in the gold and silver mills of California and other western states, San Francisco was the seat of the primary American market and San Francisco quotations governed. On account of a number of economic and technical changes and developments, the controlling American market has recently shifted to New York City, therefore the *Engineering and Mining Journal's* metal quotations now carry New York quotations primarily.

"Up to the time of the World War, the average of San Francisco quotations was used annually in computing the yearly values for Cali-

fornia's quicksilver yield in the reports of the State Mining Bureau (now Division of Mines). Owing to rapid fluctuations during the years subsequent to 1914, and the fact that quotations did not always mean sales, we adopted a policy of applying the prices actually received by the producers at Californian shipping points as the measure of our statistical values.

"A valuable part of the annual statistical reports of the Federal Bureaus (Geological Survey and Mines) is that giving data on the markets for quicksilver and the proportions used in the various industries and arts. The sources of those data are largely beyond the immediate reach of our state division to obtain directly."

DISCUSSION*

[This discussion refers also to the paper by L. H. Duschak, beginning on page 283.]

L. H. Duschak.—Mr. Maier quotes Mr. Hulings as saying that one of the serious objections to the Scott furnace is the large retention of mercury by the brick condensers, owing to the penetration of the mercury, not only into the joints between the brick but into the pores of the bricks themselves.

This is true, but it should be pointed out that this is not the fault of the Scott furnace, but of the type of condenser system that was formerly used with it. This difficulty can be entirely obviated by using a condenser system constructed of metal and tile pipes and wooden tanks.

Treatment of condenser mud will only be necessary where water sprays are used or where rather exceptional conditions lead to the production of a certain amount of low-grade mud, which cannot be easily separated by fractional settling. Material of that sort, containing free mercury, or even synthetic sulfide, is readily amenable to flotation and yields 50 to 75 per cent. concentrate.

Just a word in regard to stack losses. With a well-designed plant there need be no concern about stack loss. It should not amount to more than 10 or 15 lb. per day at the outside. The extent to which mercury is recovered from condenser gases is remarkable. A furnace handling 80 tons per day will discharge from 3000 to 5000 cu. ft. of gas per minute, which means 6 to 10 tons of gas per hour. That 6 to 10 tons of gas, with good condenser design, will not carry more than a pound or two of mercury at the outside.

The use of water sprays for condensing is undesirable when it can be avoided. It produces a mud, which has to be subsequently treated for the recovery of the metal. It is much better to operate with dry condensers whenever possible. The portion of the condenser close to the furnace becomes coated with droplets of mercury and these droplets assist in condensation by reducing supersaturation. If water sprays are used the mercury particles become filmed with water and are thus prevented from picking up additional mercury from the gas stream. The water film also interferes with the coalescing of the small globules.

H. L. Hazen, San Francisco, Calif.—If I remember correctly, Mr. Bradley said, in his book, that the recovery by furnacing ranged from 50 per cent. to 75 per cent. of the mercury originally in the ore. I have heard elsewhere that we now get recoveries to the extent of 90 per cent.

* At the same session of the San Francisco Meeting, A. V. Udell presented a paper on Mercury: Its Use and Usefulness, which appears with discussion in *Mining and Metallurgy*, December, 1929.

L. H. Duschak.—With good practice it is possible to get a recovery of 85 to 90 per cent.; under favorable conditions it might be somewhat above 90 per cent. There is no difficulty in getting complete extraction from the ore. The stack loss can be limited to a small amount and the other losses at various points in the plant, such as the mechanical escape of quicksilver through foundations, are all things that can be controlled. These miscellaneous losses should not exceed 5 per cent. My belief is that good operation ought to yield at least 90 per cent. actual recovery.

W. W. Bradley, San Francisco, Calif.—The statement referred to as having been made in my book (*Bull.* 78, California State Mining Bureau) published some eleven years ago, that recoveries were from 50 to 75 per cent., as I recall, had reference in particular to many of the old rule-of-thumb operations, when the operators were not altogether careful about some of the construction features in the maintenance of plants. I think I did say, however, that recoveries better than 90 per cent. were made where technical control was good.

Dr. Richards told us last night, you will recall, that he wrote the first two volumes of his book on ore dressing, and that between the time it went to the printer and was made available for distribution the Wilfley concentrating table was invented and made his book obsolete. A little later he prepared two more volumes, bringing the subject of ore dressing up to date, and about that time flotation came into use and rendered the rest of the books obsolete.

My experience in writing about metallurgy and ore dressing of quicksilver has been somewhat more fortunate, but I had a rather narrow escape. The bulletin had been prepared and was ready for the printer—in fact, I believe I had received the galley proofs—when Mr. Gould, at New Idria, found that his first rotary was going to be successful. He furnished me with a diagram of the plant and a photograph of the furnace in operation, and I was able to get it in just before the final page proofs. It was a rather fortunate thing for me, otherwise my book would have become obsolete overnight.

The interesting thing in that connection is that, as Mr. Duschak pointed out, there have not been, except for minor improvements in mechanical arrangements, any improvements or changes in the quicksilver metallurgy in some years. The present status of metallurgy of quicksilver is fairly satisfactory.

R. G. Hall, San Francisco, Calif.—Are the actual stack losses commensurate with the losses that might be expected from the vapor tension of mercury prevailing in the stack conditions, as they would be indicated from the temperature and pressure therein, or are those losses greater or less than that?

L. H. Duschak.—With good condenser practice the stack loss will be little more than that due to the saturation of the escaping gas with mercury vapor.

R. L. Humphreys, Berkeley, Calif.—Mr. Duschak also spoke about the lag in the condensation of mercury vapor; that is the slow release of supersaturation, which makes it necessary to use several large condensing chambers. I wonder if he tried treating the mercury vapor with a shower of atomized mercury.

L. H. Duschak.—As far as I know, that has never been tried, but various arrangements have been used to promote turbulence and to expose a greater surface of contact in order to promote that release of supersaturation.

J. H. Batcheller, Corvallis, Ore.—How long has the Cottrell method been used for dust? Is there any way by which, beforehand, there are characteristics that would help one to decide whether or not there would be a fair prospect of success with it? I was told at the Opalite mine, where it was successful, that it had been tried elsewhere without success.

L. H. Duschak.—The first use of the Cottrell in connection with quicksilver was at New Almaden in 1917 or 1918, in connection with a 16-ft. Herreshoff furnace. In answering the second question, I will give a little more detail with respect to Sulphur Bank. As far as I know, that is the only case where the Cottrell was not as successful as was expected. There were two reasons for that: (1) the quantity of dust, which was very much in excess of what had been anticipated, and (2) the character of the dust. The Cottrell process depends on the application of a unidirectional high-tension current. The wire, the discharge electrode, is the negative electrode, and the theory is that the dust particles take on this electrical charge and under the action of the electrical field are forced over to the collecting electrode, which may be the circular or square pipe. The layer of dust formed on the collecting electrode must be moderately conducting, so that the negative charge may drain rapidly to earth.

In certain other fields of metallurgy, for example in collecting zinc oxide fume, it has been found that unless the humidity is very high the zinc oxide forms a practically insulating layer and the process does not work as it should. We encountered that same thing at Sulphur Bank. Under the method of operation there all of the dust passed through the hot zone in the furnace and was baked at a temperature of 650° or 700° C., which rendered it relatively nonconducting. The dust layer that formed in the precipitator tended to reduce the efficiency.

I can say, positively, that in the ordinary countercurrent firing, as in the Scott furnace, the Herreshoff, or the usual rotary practice, the dust can be effectively precipitated by the Cottrell process. When there is a large quantity of dust the precipitator must be larger than otherwise. The Sulphur Bank experience was the only one in which the Cottrell did not perform better than was promised.

Member.—What possibility is there of discovering blind deposits of quicksilver ore?

C. N. Schuette, San Francisco, Calif.—That is a very difficult question to answer. I think it is a fact that quicksilver orebodies, certainly those of the type commonly found in California, need not necessarily outcrop, and that those thus far found were incidentally exposed to erosion.

It can be shown, I think, that the largest and richest orebodies of quicksilver the world over are due to a concentration of the primary mineralization during deposition, caused by favorable stratigraphy. In reviewing the descriptions of the largest quicksilver mines, such as Almaden in Spain, the Monte Amiata mines and Idria in Italy, the New Almaden, New Idria, Sulphur Bank, Knoxville, Oat Hill and other large mines, it is clearly evident that a concentration of the primary mineralization took place in an open-textured rock covered by a nonporous cap rock. Orebodies, as distinguished from mere mineral occurrences, were formed because of these favorable stratigraphic features.

Where the orebodies have not been exposed by erosion, the outcrops are relatively insignificant; for instance, the New Almaden mine had a very small outcrop that gave no indication of the tremendous orebodies below and at several periods of its life was in danger of being abandoned because of a supposed exhaustion of its ore. This mode of occurrence gives nearly all quicksilver mines certain distinct characteristics which aid in finding them, delineates the ore zone, and practically determines the mining method to be employed.

In California nearly all the quicksilver mines are formed in the Franciscan series of rocks, mainly sandstones, shales, and cherts. This series of rocks was at one time intruded by large masses of peridotites that eventually changed to the serpentines as we find them today. The intrusion of these peridotites shattered the Franciscan rocks in the contact area. At New Idria, for example, a peridotite laccolith is exposed for a length of some 15 miles and a width of 3 to 4 miles. The Franciscan rocks along

the contact are shattered and broken, twisted and contorted. The movement along the contact formed a fault gouge which forms an impervious barrier to solutions. At a much later period, extrusions of basalt accompanied another period of tectonic movements in California. These extrusions in general followed the planes of structural weakness along the contacts of the former peridotite intrusions. The quicksilver deposition seems to have followed shortly after the injection or ejection of the basalts, the mineralizing solutions coming up along these same planes of weakness.

Where the solutions escaped to the surface they were largely lost; they mixed with meteoric waters or escaped over the surface and no orebodies resulted. In some places, where these waters were held in basins at the surface, sinter deposits containing small amounts of cinnabar were formed. This type of deposit is more common in Nevada but minable orebodies were formed in relatively few instances. Where the rising quicksilver-bearing solutions were confined to areas of broken or shattered rock by the impervious contact gouges, orebodies were formed. At New Idria, for instance, the hanging-wall gouge curves concavely to the north and it dips south into the hill. Thus the main ore zone is a mass of shattered rock confined under an inclined impervious cap rock of inverted trough shape. This cap rock confined the mineralizing solutions to the shattered receptacle rock below it and so compelled a concentration of the deposited mineral matter.

At New Almaden the entire core of Mine Hill is one of these peridotite intrusions in Franciscan rocks. The contact is shattered and marked by a heavy gouge. This hanging-wall gouge was named the *alta* by the miners and as such a hanging-wall gouge was common to nearly all the early quicksilver mines in California, this term was used at all of them. At New Almaden, where the contact is steep, the ore deposit under it is relatively thin. Where the alta lies flat, and so effectively dammed the rising ore solutions, large rich orebodies were formed by the resulting concentration. The Santa Rita orebodies, formed under such a flat alta, supplied the furnaces for many years. The ore from them averaged 25 per cent. and produced 75,000 flasks from this one immense concentrated deposition.

North of San Francisco, at Oat Hill, Sulphur Bank and Knoxville, the concentration of the ore solutions to form orebodies took place under some of the basalt and other lava flows acting as cap rocks. The Oat Hill orebodies were exposed by erosion and were found in the regular prospecting manner. Sulphur Bank was not exposed and was found while mining for sulfur. Knoxville was exposed by erosion and was accidentally discovered while building a county road through the outcrop. These three mines are an interesting trio. At Sulphur Bank the lava cap rock is still in place and ore deposition may still be taking place. A large fault runs east and west under the lava cap and solfataric action is evident; gas and hot water are still coming up. Away from the lava cap this fault can be traced on the surface by the lack of vegetation, and on a sultry day the odor of gas is noticeable along there. Only where the basalt cap was in place was there any concentration of the primary mineralization into an orebody.

At Knoxville, at the time the ore deposit was found, the lava capping under which it had formed had been removed to some extent by erosion and this orebody had been lying exposed to the elements for no one knows how long. Due to this exposure, secondary minerals developed, such as metacinnabar and probably secondary cinnabar as well. At Oat Hill the remnant of the lava cap that once extended over it is broken up, the orebodies that were formed just under the cap are partly removed by erosion and the orebodies that were mined represent the feeder fissures to these orebodies. These fissures extend laterally from the stock or dike of the basalt which came up and covered the deposit.

Member.—Suppose that there was such a body as you describe at a depth of 500 ft., would you be able to know what was there sufficiently to go after it?

C. N. Schuette.—These lava flows, as at Oat Hill, Sulphur Bank and Knoxville, are relatively thin. Suppose you are able to find a large fault, which would be a possible source fissure for mineralizing solutions, and its course extends under a lava flow which could serve as a cap rock. If the lava capped a porous or shattered rock that would constitute a favorable receptacle rock for ore deposition, I think there would be a fair chance of finding an orebody concentrated beneath it. There are many such lava flows around Mount St. Helena, for example, and faults are also in evidence. The known quicksilver mines are at the edges of these lava flows where the orebodies have been exposed by erosion. There is no reason why blind orebodies should not exist and a geological study of conditions followed by drilling of the cap rock flows, which are from 20 to 100 ft. thick, or more, would seem a fair prospecting gamble at certain localities.

Member.—Are discoveries of that sort made under those conditions?

C. N. Schuette.—Yes and no. Discoveries have been made in this way by drilling for probable extensions of known orebodies by postulating its formation on this general theory. New orebodies have been found in this way, particularly in Texas. No one, of course, has gone out to drill, say, an anticline in a nonmineral region where it was probable that a sandstone was overlain by a shale, in the hope of striking a cinnabar orebody, but orebodies have been found in searching for geological structures that would favor a concentration of the primary mineralization in regions where quicksilver is known to exist, even though no commercial deposits had been developed.

L. H. Duschak.—Is there any more reason for expecting blind deposits of cinnabar than of any other mineral?

C. N. Schuette.—I can not say what the comparison would be.

L. H. Duschak.—May we not expect to find blind deposits of copper, for instance? Have we not records like that?

C. N. Schuette.—Probably. Certainly silver-lead orebodies have been formed and found in the same general way. The point I am trying to make is that all orebodies of quicksilver are due to a concentration of the primary mineralization while those of other ores may be due to other causes.

Member.—In Tehachapi Pass the formation generally predominated and quicksilver also predominated. Is there any characteristic there to justify further explorations in that sort of country?

C. N. Schuette.—There are a number of deposits of that general type in Oregon and at least one in Nevada. These generally form low-grade ore deposits because there is no porous receptacle rock in them. Small local concentrations occur in this deposit near cross fissures, but my remarks apply in general only to the finding of large high-grade orebodies or large-scale concentrations and I do not know that the Tehachapi deposit could be under this heading.

In looking for quicksilver orebodies, one should look for a stratigraphy that would favor a concentration of the primary mineralization during deposition. Certain other ores are also, at times, formed under these same general conditions, there is no doubt of that, but I do think that all large quicksilver orebodies are formed in such a way, and a recognition of that fact will enable us to search intelligently for blind orebodies of quicksilver in certain quicksilver-bearing regions.

Progress in Production and Use of Tantalum

By George W. Sears,* Reno, Nevada

(New York Meeting, February, 1930)

Until a comparatively few years ago, interest in tantalum was limited almost wholly to its scientific investigation, but its extreme resistance to the action of even the strong mineral acids, its great avidity for the common gases, hydrogen, oxygen and nitrogen, and its capacity for transforming alternating to direct current are winning for it a place in our industrial life.

Tantalum is not attacked by any single acid except hydrofluoric, nor is it affected by solutions of alkalies. Aqua regia, which readily dissolves both platinum and gold, has no action on it. This resistance to the common corrosive agents is an outstanding property of tantalum and accounts for many of the applications to which it has been put. When heated, tantalum reacts readily with oxygen and nitrogen and absorbs large volumes of hydrogen, 740 volumes being taken up at a dull red heat. This capacity for absorbing gases is being utilized to an increasing extent in the manufacture of vacuum tubes, in which tantalum is used for the metal parts, especially for grid wire. It is said to be a superior metal for this purpose, particularly in power and sending tubes, because of its low secondary emission.

The use of tantalum in the manufacture of electrodes, dishes, spatulas and other laboratory apparatus appears to be fairly steady.

There has been some development in its use for dental instruments, surgical tools, pen points, hypodermic needles and acid-proof pumps. Recently it has been used in the manufacture of spinnerets for the rayon industry, of nozzles for the manufacture of artificial casings for sausages and of lining for plant equipment.

Tantalum in Electrolysis

It has been known for some time that certain substances permit the passage of an alternating current in one direction more readily than in the other. Tantalum possesses this property to a marked degree.[1] When an alternating current is passed through a cell with tantalum electrodes and sulfuric acid electrolyte, a thin film is soon formed at the

* Head of Department of Chemistry, University of Nevada.

[1] E. W. Engle: Tantalum, Tungsten and Molybdenum. *Trans.* A. I. M. E. (1925) **71**, 698.

pole and the current stops. If lead replaces one of the tantalum electrodes, the current will flow from the lead to the tantalum but not from the tantalum to the lead. This property has been utilized in the manufacture of electrolytic rectifiers, originally used for charging radio batteries but now used chiefly in connection with railways for charging signal batteries. Tantalum cathodes have proved satisfactory for the electrolytic determination of such metals as silver, platinum, copper, zinc and nickel, since these metals can be dissolved from it by acids or aqua regia.

Tantalum Alloys

Tantalum as an alloy material has received considerable study and many alloys have been reported. While pure tantalum is relatively soft and malleable, it may be greatly hardened by alloying with small amounts of other substances. Apparently few of these alloys have found their way into industrial uses. Among them, three types appear to be most prominent—alloys of high melting point, hard alloys and acid-resisting alloys. The alloys of high melting point contain from 1 to 40 per cent. tantalum, together with varying proportions of such other elements as molybdenum, tungsten, carbon, manganese, nickel, vanadium and silicon. It is claimed that in some of these tungsten or molybdenum may be replaced wholly or in part by tantalum. As much as 50 per cent. tantalum is put into the hard alloys recommended chiefly for the manufacture of tools, which contain also a variety of other elements; *i. e.*, iron, chromium, vanadium, tungsten, molybdenum and carbon. Except for use as dental and surgical instruments, pen points, etc., in which the quantity of metal used is relatively small, tantalum is as yet too expensive to hope for much greater development in this direction.

The proportion of tantalum necessary to the acid-resisting and corrosion-resisting alloys appears to be relatively small. The first rustless steel was an alloy of tantalum and iron. Since its introduction, however, other alloys of this type have been developed and the use of tantalum in this capacity is meeting with considerable competition. A few new alloys of tantalum with iron and with aluminum have been developed during the past few years.

It is probable that the greater proportion of the tantalum that is being produced goes into the manufacture of products that require the pure metal. Perhaps one reason for this is found in the difficulties involved in its extraction and metallurgy, which make it more expensive than many of the other metals with which it must compete.

Tantalum Ores and Production

Tantalum must be looked upon as a rare metal. It has been estimated that the actual quantity of tantalum in the earth's crust is less

than that of gold.[2] Australia continues to be the greatest producer of high-grade tantalum ore, though the Black Hills of South Dakota produces some columbite carrying 18 to 40 per cent. Ta_2O_5. Some tantalite, running over 80 per cent. Ta_2O_5, is also found in the Black Hills.[3] Reports from Colorado[4] indicate the presence in that state of a considerable deposit of tantalite averaging 50 per cent. Ta_2O_5. Samarskite, found in Brazil,[5] contains about 14 per cent. Ta_2O_5. These last two deposits, however, have not been worked for tantalum. Until 1928, when the production rose to nearly 35,000 lb., relatively small amounts have been mined in the United States during the last decade. The largest amount, 4000 lb., was produced in 1920. Imports of some 15,000 lb. of ferroalloy were reported for 1927; much more than were reported in any previous year.[6] While this decided increase in production points to a greater interest in tantalum products, the indications are that the local production at least will be much smaller for 1929.[7] This decrease may be made up by an increased importation, because some of the greatest users are getting all of their ore from the Australian tantalite.

Extraction and Metallurgy of Tantalum and Columbium

The extraction and metallurgy of tantalum are difficult chemical processes. There is but one type of mineral available, and this contains both tantalum and columbium. Where the tantalum predominates it is called tantalite and where the columbium predominates it is called columbite. Since columbium has practically no commercial importance at the present time, it serves only to increase the cost of the tantalum, hence the desire of the manufacturer for a high-grade tantalite. For this reason the Australian tantalite, carrying 60 per cent. or more of Ta_2O_5, is generally preferred to the columbite found in the Black Hills.

The present method of extraction involves an alkali fusion and subsequent treatment with hydrofluoric acid and potassium fluoride and the reduction of the double fluoride in vacuum furnaces. This separates the greater part of the tantalum as K_2TaF_7. The remainder is left with the columbium and can be separated from it only by difficult, tedious and time-consuming procedure. Undoubtedly this difficulty in obtaining pure columbium material has caused its commercial development to lag behind that of tantalum. Its preparation in the pure state

[2] C. W. Balke: Metals of the Tungsten and Tantalum Groups. *Ind. & Eng. Chem.* (1929) **21,** 1002.

[3] Columbite-Tantalite. South Dakota School of Mines *Bull.* 16 (1929) 255.

[4] Colorado Management for Vanadium Properties. *Eng. & Min. Jnl.* (1927) **124,** 462.

[5] H. G. Thut: Rare-metal Deposits in Brazil. *Chem. Absts.* (1928) **22,** 3377.

[6] Cobalt, Molybdenum, Nickel, Tantalum, etc. Min. Resources of U. S. for 1928

[7] Personal communication.

[8] C. W. Balke: *Op. cit.*

has become possible only recently. No data could be obtained as to the method used, but it is probably similar to that used for tantalum. A more recent development claims[9] that both tantalum and columbium can be obtained free from hydrogen and nitrogen by heating their oxides, in sealed or evacuated containers, with calcium, calcium chloride and an alkali metal.

Greater simplicity in the extraction and metallurgy of these two elements and the development of a market for columbium would undoubtedly be of great benefit to the tantalum industry and make for a greater use of tantalum products. It is reported[10] that during a part of the year 1928 purchasers of the South Dakota columbite paid for the columbium content as well as for that of tantalum, which indicates that some interest is being given to columbium. A few years ago[11] it was found, in the laboratory of the University of Nevada, that a complete separation could be made between tantalum and columbium by fusion of their ores with sodium pyrosulfate at a temperature of 835° to 875° C. Under these conditions the columbium can be completely dissolved in strong sulfuric acid, while the whole of the tantalum remains in the insoluble residue. Since then the method has been developed into a quantitative determination[12] of the two elements, and work is now in progress on the application of this principle on a larger scale. Although it is too early to predict what the outcome will be, there is some indication that it can be accomplished. If so, it will greatly simplify the extraction and purification of both tantalum and columbium. It can hardly be hoped that the extraction and metallurgy of either tantalum or columbium will ever be made as simple as that of many of the other rarer metals. The great similarity in the chemical properties of these two elements, the acidic nature of their compounds, their extreme inactivity toward the more common reagents and their strong tendency to remain in chemical combination, all unite to make their production both difficult and expensive.

DISCUSSION

D. M. Liddell, New York, N. Y.—Mr. Smyth, do you happen to know, or does anyone, the way this columbium is produced?

W. I. Smyth, Reno, Nev.—I asked Dr. Sears about that. He said he had been trying to get data on the subject but they are difficult to obtain. Dr. Sears said that there must have been some demand for columbium at some time, because for a while, either in 1928 or 1929, the South Dakota producers were being paid for the columbium content.

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[9] J. W. Marden: Production of Rare Metals. U. S. Patent 1728941 (Sept. 24, 1929)

[10] Andalusite-Sillimanite. South Dakota School of Mines *Bull.* 16 (1929) 257.

[11] G. W. Sears: Critical Studies on the Fusion of Rare Metal Ores.—II. The Separation of Tantalum and Columbium. *Jnl.* Amer. Chem. Soc. (1926) **48**, 343.

[12] G. W. Sears: Critical Studies on the Fusion of Rare Metal Ores.—III. Determination of Tantalum and Columbium. *Jnl.* Amer. Chem. Soc. (1929) **51**, 122.

P. M. Tyler, Washington, D. C.—I have one suggestion about the use of columbium. It came to my attention, as Dr. Sears pointed out, that in the case of some of the South Dakota material both the Cb_2O_5 and the Ta_2O_5 were being paid for. I made some inquiries in New York, and it seemed to be the impression that some of the ore had been sold to England for the manufacture of ferrotantalum for use in high-speed steel. Whether that is true or not, I do not know, but I offer the suggestion.

D. M. Liddell.—So you would think there was use for a metal which was actually a mixture of ferrotantalum and ferrocolumbium?

P. M. Tyler.—That is what I was told in England. I was informed unofficially that columbium and tantalum were interchangeable in these high-speed steel alloys. Whether or not it is true, I do not know, but I understood that commercially it was so considered. But the fact that in 1929 there was no market for columbium content indicates that something happened. I am extremely curious to find out what it was.

I have been shown specimens of columbite or tantalite, I do not know which, which were reported to have come from North Carolina. These were only specimens and I do not know whether the occurrence can be developed commercially or not.

G. W. Sears (written discussion).—In explanation of the use of Cb_2O_5 that was sold during 1928, a personal letter from Prof. F. C. Lincoln, of the South Dakota School of Mines, states that the shipments went mainly to England, where it was said the material was used for machine-shop drills. The rise in price of tantalum, coupled with the improvement in the tungsten-carbon alloys for the same use, resulted in a complete falling off in demand this summer (1929).

Contents of 1930 Volumes

TRANSACTIONS, Institute of Metals Division, 1930. 599 pages. Index. Papers presented before Institute of Metals Division at Cleveland, Sept. 9–12, 1929 and New York, Feb. 17–20, 1930.

Papers as follows: Hard Metal Carbides and Cemented Tungsten Carbide (Annual Lecture), by Samuel L. Hoyt. COPPER AND BRASS: Directed Stress in Copper Crystals, by C. H. Mathewson and Kent R. Van Horn; Thermal Conductivity of Copper Alloys, I—Copper-Zinc Alloys, by Cyril Stanley Smith; Causes of Cuppy Wire, by W. E. Remmers; Certain Types of Defects in Copper Wire Caused by Improper Dies and Drawing Practice, by H. C. Jennison; Correlation of the Ultimate Structure of Hard-drawn Copper Wire with the Electrical Conductivity, by R. W. Drier and C. T. Eddy; Deoxidation of Copper with Calcium and Properties of Some Copper-calcium Alloys, by Earle E. Schumacher, W. C. Ellis and John F. Eckel; Alpha-phase Boundary of the Ternary System Copper-silicon-manganese, by Cyril Stanley Smith; Alpha-beta Transformation in Brass, by Albert J. Phillips; Note on the Crystal Structure of the Alpha Copper-tin Alloys, by Robert F. Mehl and Charles S. Barrett; Eutectic Composition of Copper and Tin, by G. O. Hiers and G. P. DE FOREST. CORROSION: Corrosion of Alloys Subjected to the Action of Locomotive Smoke, by F. L. Wolf; Internal Stress and Season Cracking in Brass Tubes, by D. K. Crampton; Stress-corrosion Cracking of Annealed Brasses, by Alan Morris. SECONDARY METALS: Reclaiming Non-ferrous Scrap Metals at Manufacturing Plants, by Francis N. Flynn; Recovery of Waste from Tin-base Babbitting Operation, by P. J. Potter; Manufacture of Wire Bars from Secondary Copper, by W. A. Scheuch and J. Walter Scott; Utilization of Secondary Metals in the Red Brass Foundry, by H. M. St. John. MELTING AND CASTING METALS: Oxides in Brass, by O. W. Ellis; Distribution of Lead Impurity in a Copper-refining Furnace Bath, by J. Walter Scott and L. H. DeWald; Comparison of Copper Wire Bars Cast Vertically and Horizontally, by J. Walter Scott and L. H. DeWald; A Theory Concerning Gases in Refined Copper, by A. E. Wells and R. C. Dalzell; Effects of Oxidation and Certain Impurities in Bronze, by J. W. Bolton and S. A. Weigand; Influence of Silicon in Foundry Red Brasses, by H. M. St. John, G. K. Eggleston and T. Rynalski; Melting Bearing Bronze in Open-flame Furnaces, by Ernest R. Darby; Recent Developments in Melting and Annealing Non-ferrous Metals, by Robert M. Keeney; Melting and Casting Some Gold Alloys, by Edward A. Capillon; Monel Metal and Nickel Foundry Practice. By E. S. Wheeler. PROPERTIES OF METALS: Effects of Cold Working on Physical Properties of Metals, by R. L. Templin; Effect of Alloying on the Permissible Fiber Stress in Corrugated Zinc Roofing, by E. A. Anderson; Effect of Heat Treatment on Properties and Microstructure of Britannia Metal, by B. Egeberg and H. B. Smith; Expansion Properties of Low-expansion Fe-Ni-Co Alloys, by Howard Scott; Metallography of Commercial Thorium, by Edmund S. Davenport; Working Properties of Tantalum, by M. M. Austin. X-RAY INVESTIGATIONS: Determining Orientation of Crystals in Rolled Metal from X-ray Patterns Taken by Monochromatic Pinhole Method, by Wheeler P. Davey, C. C. Nitchie and M. L. Fuller; X-ray Notes on the Iron-molybdenum and Iron-tungsten Systems, by E. P. Chartkoff and W. P. Sykes. SYSTEM CADMIUM-MERCURY: The System Cadmium-mercury, by Robert F. Mehl and Charles S. Barrett.

TRANSACTIONS, Coal Division, 1930. 724 pages. Index. Papers presented at New York, February, 1928, February, 1929 and February, 1930.

Papers as follows: MINING: Ventilation Problems at the World's Largest Coal Mine, by Henry F. Hebley; Coal-mining Operations in the Sydney Coal Field, by A. L. Hay; Barrier Pillar Legislation in Pennsylvania, by George H. Ashley; Subsidence from Anthracite Mining, by H. W. Montz, With an Introduction on Surface Support by R. V. Norris; The Royal Commission on Mining Subsidence, by Henry Louis; Subsidence in Thick Freeport Coal, by John M. Rayburn; Bumps in No. 2 Mine, Springhill, Nova Scotia, by Walter Herd; Misfires in Anthracite Coal Mines, by T. D. Thomas; Misfires in Bituminous Coal Mines, by W. H. Forbes; Misfires in Non-metallic Mining (Limestone), by A. W. Worthington. CLEANING: Hindered-settling Classification of Feed to Coal-washing Tables, by B. M. Bird and H. F. Yancey; Re-treating Middlings from Coal-washing Tables by Hindered-settling Classification, by B. M. Bird and H. F. Yancey; Coal Washability Tests as a Guide to the Economic Limit of Coal Washing, by George Stanley Scott; Cleaning Bituminous Coal, by J. R. Campbell. COKING: Test for Measuring the Agglutinating Power of Coal, by S. M. Marshall and B. M. Bird. Loss in Agglutinating Power of Coal Due to Exposure, by S. M. Marshall, H. F. Yancey and A. C. Richardson; Relation of By-product Coke Oven to the Natural Gas Supply of the Pittsburgh District, by H. J. Rose. CLASSIFICATION: Introduction, by A. C. Fieldner; Classification of Coal from Proximate Analysis and Calorific Value, by W. T. Thom, Jr.; Classification of Coals by Ultimate Analysis, by H. J. Rose; Classification of Coal from the Viewpoint of the Geologist, by M. R. Campbell; Classification of Coal from the Viewpoint of the Paleobotanist, by R. Thiessen; Classification of Coal in the Light of Recent Discoveries with Regard to Its Constitution, by W. Francis; Commercial Classification of Coal, by F. R. Wadleigh; Classification of Coals from the Point of View of the Railroads, by E. McAuliffe and M. Macfarland; Use Classification of Coal as Applied to the Gas Industry, by W. H. Fulweiler; Classification from the Standpoint of the By-product Coke Industry, by W. H. Blauvelt; Classification of Coal from the Standpoint of the Steam Power Consumer, by S. B. Flagg; Classification

of Coal from the Standpoint of the Coal Statistician, by F. G. Tryon; Closer Cooperation Between Scientists and Practical Men (Round Table Discussion); Natural Groups of Coal and Allied Fuels, by M. R. Campbell; Coal Classification; a Review and Forecast, by George H. Ashley; Outline of a Suggested Classification of Coals, by David White; Status of Coal Classification in Canada, by R. E. Gilmore; Multibasic Coal Charts, by H. J. Rose; Changes in Properties of Coking Coals Due to Moderate Oxidation during Storage, by H. J. Rose and J. J. S. Sebastian; Review of Methods Used in Coal Analysis, with Particular Reference to Classification of Coal, by A. C. Fieldner; Present Status of Ash Corrections in Coal Analysis, by A. C. Fieldner and W. A. Selvig; Determination of Mineral Matter in Coal and Fractionation Studies of Coal, by E. Stansfield and J. W. Sutherland; Constitution and Nature of Pennsylvania Anthracite with Comparisons to Bituminous Coal, by Homer Griffield Turner; Splint Coal, by Reinhardt Thiessen; Commercial Classifications of Coal, by F. R. Wadleigh; Commercial Descriptions of Pennsylvania Anthracite, by E. W. Parker; Properties of Coal Which Affect Its Use for the Manufacture of Coal Gas, Water Gas and Producer Gas, by Gilbert Francklyn.

Transactions, Petroleum Development and Technology, 1930. 610 pages. Index. Papers presented before Petroleum Division at Tulsa, Oct. 3–4 and Los Angeles, Oct. 4–5, 1929 and New York, Feb. 18–20, 1930.

Approximately 70 papers, as follows: Unit Operation of Oil Pools: Committee Reports: Genera, Summary, by Unitization Committee; Eastern United States and Foreign Countries, by H. H. Hill and E. L. Estabrook; Oklahoma and Kansas, by A. W. Ambrose and C. E. Beecher; Arkansas, Louisianal Texas and New Mexico, by F. H. Lahee; Rocky Mountain Region, by F. E. Wood; Salt Creek, by Rocky Mountain Unitization Committee; Rock River, by Wilson B. Emery; Hiawatha and Baxter Basin, by W. T. Nightingale; Hidden Dome, by Wilson B. Emery; California, by Joseph Jensen; Principles of Unit Operation, by Earl Oliver and J. B. Umpleby; Some Developments and Operating Economies of Unit Operation, by Sam Harlan; Suggested Procedure for Exploitation of an Oil-bearing Structure by Unit Operation, by C. S. Corbett. Production Engineering: Summary, by C. V. Millikan; Theory of Well Spacing, by W. P. Haseman; Spacing of Wells in the Long Beach Field, by Dwight C. Roberts and Stender Sweeney; Well Spacing in the Salt Creek Field, by F. E. Wood; Equilateral Triangular System of Well Spacing, by C. S. Corbett; Quantitative Effect of Gas-oil Ratios on Decline of Average Rock Pressure, by Stewart Coleman, H. D. Wilde, Jr. and Thomas W. Moore; Condensation Effect in Determining Gas-oil Ratio, by Alexander B. Morris; Mathematical Development of the Theory of Flowing Oil Wells, by J. Versluys; Flow Resistance of Gas-oil Mixtures through Vertical Pipes, by L. C. Uren, P. P. Gregory, R. A. Hancock and G. V. Feskov; Some Observations on Principles Involved in Flowing Oil Wells, by S. F. Shaw; Classification of Flowing Wells with Respect to Velocity, by F. P. Donohue; Mid-Continent Practices in Handling Flowing Wells, by Reid W. Bond, D. L. Trax, C. D. Watson and Morgan Walker; Repressuring in the Selover Zone at Seal Beach and the Effect of Proration, by A. Hamilton Bell and E. W. Webb; Repressuring in Depleted Oil Zones, by C. M. Nickerson; Modern Practice in Water-flooding of Oil Sands in the Bradford and Allegany Fields, by Paul D. Torrey; Mechanics of a California Production Curve, by Stanley C. Herold; Methods of Tubing High-pressure Wells, by H. C. Otis; Deep Sand Development at Santa Fe Springs, by Joseph Jensen, McDowell Graves, W. D. Goold and M. L. Gwin. Research: Recent Studies on the Recovery of Oil from Sands, by Joseph Chalmers; Law of Flow for the Passage of a Gas-free Liquid through a Spherical-grain Sand, by William Schriever; Variation of Pressure Gradient with Distance of Rectilinear Flow of Gas-saturated Oil and Unsaturated Oil through Unconsolidated Sands, by W. F. Cloud; Behavior of Gas Bubbles in Capillary Spaces, by Ionel I. Gardescu; Cementing Problem on the Gulf Coast, by H. D. Wilde, Jr.; Drilling Mud Practice in the Ventura Avenue Field, by F. W. Hertel and E. W. Edson; Review of Oil-field Corrosion Problems for 1929, by L. G. E. Bignell. Economics: Summary, by Warren A. Sinsheimer; Economic Trend of the Petroleum Situation, by Joseph E. Pogue; Controlled Gasoline Supply—the Key to Oil Prosperity, by H. J. Struth; Problems of Petroleum, by J. Elmer Thomas; Influence of Control in the Oil Industry upon Investment Position of Oil Securities, by Barnabas Bryan. Production: Summary, by C. P. Watson; Kansas, 1928 and 1929, by Charles E. Straub and Anthony Folger; Oklahoma, by H. B. Goodrich; West Texas and Southeast New Mexico, by R. E. Rettger; East Texas and Along the Balcones Fault Zone, by F. E. Poulsen; North Central and West Central Texas, by J. W. Lewis; Southwest Texas, by O. G. Bell, Texas Panhandle, by W. E. Hubbard; Gulf Cost of Texas and Louisiana, by R. H. Goodrich; Arkansas, by H. W. Bell; California, by D. B. Myers; Rocky Mountain District, by F. F. Hintze; North Rocky Mountain Region, Including Wyoming, Montana and Alberta, by Ralph Arnold and O. I. Deschon; Appalachian Fields, by Charles R. Fettke; Indiana and Illinois, by Alfred H. Bell and Paul F. Simpson; Mississippi, by R. E. Grim; World Production during 1929, by Valentin R. Garfias; Venezuela, by E. L. Estabrook and J. A. Holmes; Russia, by B. B. Zavoico; Mexico, by Valentin R. Garfias and C. O. Isakson; Dutch East Indies and Sarawak (Western Borneo), by J. Th. Erb; Rumania, by Special Correspondence; Colombia, by Michael O'Shaughnessy; Argentina, by José M. Sobral; Canada, by T. G. Madgwick and W. Calder; Bolivia, by G. P. Moore. Refining: Petroleum Refining Summary, by A. D. David. Engineering Education: Summary, by H. C. George.

ABSTRACTS

ON the following pages are abstracts of papers published by the Institute during the year 1930 as *Technical Publications,* preprints, in bound volumes and as leading articles in MINING AND METALLURGY.

Many of the *Technical Publications* have been reprinted in bound volumes. Information regarding this disposition, and number of pages in each paper, may be found in the list beginning on page 419.

The abstracts are grouped as follows:

METAL MINING

Miami Copper Company Method of Mining Low-grade Orebody

By F. W. MACLENNAN, Miami, Ariz.

(Tech. Pub. No. 314; Class A, Metal Mining, No. 34)

THE high-grade orebody at Miami was mined successively by top-slicing, shrinkage, stoping and under caving. The method described in this paper was developed to enable the low-grade orebody (36,000,000 tons assaying 1.06 per cent copper) to be mined at a low cost without further dilution of the grade. The method is clearly described and illustrated by drawings and photographs, and the original paper should be consulted as it is impracticable to abstract it, beyond noting that the mining cost is just under 40 cents per ton and the average grade of ore now being produced yields less copper than the mill tailing of 1915, the tons mined per day having increased from 4000 to 18,000 in the same period.

Vertical and Incline Shaft Sinking at North Star Mine

By ARTHUR B. FOOTE, Grass Valley, Calif.

(Tech. Pub. No. 324; Class A, Metal Mining, No. 36)

THE main shaft of the North Star mine is an inclined one following the main vein at an average dip of 26 deg., a vertical shaft intersects it at the 4000-ft. (incline) level, the incline continuing to the 6300-ft. level. At this point the vein was found to give out and a new vein, called the No. 1, was found to dip in exactly the opposite direction.

After studying the problem it was decided to sink an inclined shaft on the new vein and deepen the vertical shaft approximately 2000 ft. to where it would intersect the incline. This work of deepening could not be allowed to interrupt hoisting through the existing part of the shaft. How the work was planned and executed is described in the original paper, which should be read in full. An average advance of 125 ft. per month, at a cost of $80 per foot, plus a charge of $20 per foot for "overhead," the contract price being $65 per foot for labor and supplies; the final total cost was $118.38 per foot. The methods and costs of sinking the inclined shaft are also described. A notable feature of sinking the inclined shaft is that 17,000 ft. of drifts were driven and 242,900 tons of ore mined during the two-year shaft-sinking period. The use of a blasting set in the vertical shaft saved 2 or 3 hours of time each round by enabling the timbermen to work while the mucking was going on and by removing the necessity for cleaning loose rock off the timbers. No fatal accident occurred in the course of the work.

Shaft Sinking at Texas Salt Mine

By M. Taylor, Pittsburgh, Pa.

(Min. & Met., December, 580. 3000 words)

SINKING in difficult conditions made possible by precementation of strata.

Mining Methods and Costs at the Presidio Mine of the American Metal Co. of Texas

By Van Dyne Howbert, New York, N. Y., and R. Bosustow, Shafter Texas

(Tech. Pub. No. 334; Class A, Metal Mining, No. 39)

THIS lead-silver mine has much interest because it is the only successful one in its district (the Big Bend region of Texas), and has been working since 1880, having now reached a depth of 500 ft. Although the title of the paper is methods and costs, the characteristics of the orebodies and wall rocks, methods of prospecting, exploration, sampling, and estimating tonnage are described in some detail. Then methods of mining are taken up, followed by transportation; wage and contract systems are described, also the surface plant. The paper concludes with four tables giving costs of the various operations.

Top Slicing With Filling of Slices, as Used at the Charcas Unit of the Compania Minera Asarco, S. A.

By Howard Willey, El Paso, Texas

(Tech. Pub. No. 364; Class A, Metal Mining, No. 41)

AFTER giving the physical and geological conditions that govern mining in this property, the author describes former mining methods and then takes up the filled top slice in detail, outlining the development work, the opening of the sill floor, slice preparations and slicing and filling. Some of the important details of the procedure are described

in detail, among them head blocks, ore and waste passes, and general precautions. The paper concludes with a summary of costs. An advantage claimed for the method is that it is easily reduced to simple routine operations that are easily learned by unskilled workmen.

Development of Benguet Mining District

By C. M. Eye, San Francisco, Cal.
(Min. & Met., November, 522. 3000 words)

DESCRIPTION of district in north central Luzon with Baguio as center; readily accessible; healthful climate; plentiful supply of pine timber. Elevation of mines, around 3000 ft. Labor, native, requiring white supervision. Power from hydroelectric plants with stand-by Diesels. Ore occurrences generally quartz and pyrite in fissure veins in andesite, carrying good values. Metallurgy, fine grinding and cyanidation with recoveries better than 90 per cent. Annual gold output in excess of $2,500,000, mostly from Benguet Consolidated and Balatoc properties. Recent developments indicate continuance of values with depth, particularly in the Consolidated mine. Increased output prospects are good.

Timbering at the Hecla Mine

By Alexander S. Corsun
(Min. & Met., August, 382. 2350 words)

A PRIZE-WINNING essay by an undergraduate at the Oregon School of Mines, describing the timbering at a mine along a nearly vertical shear zone in quartzite with a substantial gouge and lamprophyre dike occurring in an irregular manner throughout the lode.

Some Recent Developments in Open-pit Mining on the Mesabi Range

By Earl E. Hunner, Duluth, Minn.
(Tech. Pub. No. 333; Class A, Metal Mining, No. 38)

THIS valuable paper has three subtopics. The first is a concise historical review of the development of open-pit mining on the Mesabi Range. The second describes significant new developments, as illustrated by methods and equipment at the Mesabi Chief. The third similarly describes the reequipment of the Susquehanna pit, the most interesting being the account of how the ore in the bottom of the pit is being hoisted through a shaft with automatic equipment, being handled to the shaft by motorized dump cars.

One of the most significant developments is the general use of electric power in the place of steam. This applies not only to hoisting, pumping, air-compressing equipment and underground haulage, but also to electric shovels used for open-cut mining, and electric locomotives for operation in the pits. In the new screening and washing plants, electricity likewise is the source of power. The motors are remote-control through push-button stations. The feature of the motorized locomotive car is that when

the locomotive is pulling a train of loaded cars, its own load of ore gives it the necessary weight to provide traction. The economy arises in eliminating the unnecessary weight when the empty train is being returned.

Several factors combine to make the well known "milling" system of mining impractical at the Susquehanna pit. One is the stickiness of the ore, which would tend to hang up in chutes, and another is the fact that inclusions of waste are frequent in the pit, and the present method makes it possible to keep different classes of material separate.

Observation on Ground Movement and Subsidences at Rio Tinto Mines, Spain

By Robert E. Palmer, London, England

(Tech. Pub. No. 271; Class A, Metal Mining, No. 30; Class F, Coal and Coke, No. 29)

Effects of excavations on overlying or adjacent ground can be classified as follows: (1) Movements due to "open-cut" excavations alone, (2) movement due to underground excavations and (3) movement due to a combination of both. This paper presents plans and sections illustrating the phenomena appearing in the Rio Tinto mines as a consequence of various kinds of excavation. The author concludes that movements do not appear to follow any recognized law, and that they frequently are quite different from what would be expected in theory. He submits the data and observations in the hope that they will add something to the sum total of the knowledge on the subject.

Ground Movement and Subsidence, 1929

By George S. Rice, Washington, D. C.

(Min. & Met., January, 20. 3000 words)

A critical review of recent literature, both here and abroad, dealing with the technical and legal problems involved in surface subsidence. Discussion, April, 224.

Ground Subsidence at Sour Lake, Tex.

By E. H. Sellards, Austin, Texas

(Min. & Met., August, 377. 3800 words)

Description of observations on the earth movement incident to the subsidence at the Sour Lake salt-dome oil field in Texas, followed by a report of the oral discussion.

Development and Installation of the Hawkesworth Detachable Bit

By Chauncey L. Berrien, Butte, Mont.

(Tech. Pub. No. 274; Class A, Metal Mining, No. 31; Class C, Iron and Steel No. 44)

The Hawkesworth detachable bit is a piece of steel weighing a few ounces that by means of tapered grooves can be easily attached to a drill shank to provide the cutting edge. The economy lies in the fact that only the bits need be carried to the surface for sharpening and the

greater drilling efficiency gained by using properly conditioned and sharpened bits only. One comparison at the Badger State mine showed over a six-month period a cost of $125,866 for regular steel compared with $8,583 with Hawkesworth bits. The Anaconda Copper Mining Co. has standardized on the Hawkesworth equipment after a long detailed study and application of it to all operating requirements, and the mining department staff is satisfied that it is safer, more efficient and cheaper than regular drill steel.

Drill Sampling and Interpretation of Sampling Results in the Copper Fields of Northern Rhodesia

By H. T. Matson and G. Allan Wallis, N'Dola Northern Rhodesia, Africa

(Tech. Pub. No. 373; Class A, Metal Mining, No. 42)

The need for outlining the orebodies, which take the form of finely disseminated copper sulfides in sedimentary rocks, by widely spaced drill holes, led to much experimental work being done to ensure reliable sampling and its correct interpretation. Diamond drills, shot drills and churn drills were used and the method of taking samples in each case is described in detail.

The importance of sludge recovery in diamond and shot drilling is emphasized and the factors controlling it, such as complete water returns, the velocity of the circulating water in the hole and the length of time taken for the sludge to reach surface, are discussed and the conclusion is reached that unless water return is complete or nearly so, sludge samples are unreliable and should not be combined with core samples.

Several formulas for combining core and sludge results on the basis of weight, volume or length are given and their use discussed.

In the case of churn drilling the details of a Jones type sample splitter are given and the method used in drilling to overcome the difficulty of recovering fine slime is outlined.

Prospecting with the Long-hole Drill in the Tri-State Zinc-lead District

By W. F. Netzeband

(Min. & Met., June, 295. 2300 words)

Summary of the experience of the operators of the district made up from replies to a questionnaire, indicating that on the whole this type of machine has given satisfactory results in prospecting.

Improved Drill Shop Equipment at Morenci Branch of Phelps Dodge Corporation

(Min. & Met., March, 186. 1200 words)

Description of several new types of machines developed at the Morenci branch of Phelps Dodge Corpn.; time and labor-saving features.

Good Organization Is Making Records at the Hooper Tunnel

By W. F. Boericke, New York, N. Y.
(Min. & Met., January, 26. 600 words)

Brief description of driving an 8 by 9-ft. tunnel through Wallace quartzite, which, save for the first 100 ft., does not require timbering, using two shifts, each making a complete cycle of drilling and mucking.

Fan Selection for Metal Mine Ventilation

By N. L. Alison, Denver, Colo.
(Min. & Met., February, 111. 2600 words)

This article points out some of the important considerations that influence the selection of fans for primary and secondary ventilation. Discussion in August, 395.

Protective Measures Against Gas Hazards at United Verde Mine

By Oscar A. Glaeser, Jerome, Ariz.
(Tech. Pub. No. 276; Class A, Metal Mining, No. 32)

Because of the high-sulfur content of the ore in the United Verde mine at Jerome, Ariz., blasting is an extremely hazardous operation. Aside from the danger arising from local dust explosions and the consequent generation of dangerous gas to men in the mine when the blasting is done there is the possibility of mine fires originating in the square set timbers. Various measures have been taken to minimize the hazard. These include: the selection of the most suitable explosives; relegation of blasting in heavy sulfide stopes to the end of the day shift; careful attention to ventilation throughout the mine; the provision of safety chambers; and careful patrol of the mine workings. As a consequence of careful attention to these points accidents arising from gas have not occurred since August of 1926.

Operation of Pressure Fans in Series

By Walter S. Weeks and Vitaly S. Grishkevich, Berkeley, Cal.
(Tech. Pub. No. 339; Class A, Metal Mining, No. 40; Class F, Coal and Coke, No. 36)

This paper begins with a mathematical discussion of the operation of fans in series and then describes a series of tests undertaken to check the theoretical analysis. The results are shown graphically.

By-passing Water into Air Lines for Fire Protection

(Min. & Met., January, 55. 400 words)

While the laying of water lines in some sections of the mine may not be warranted, in most metal mines it is necessary to lay an air line to all working places. In the mine of the United Verde Extension Mining Co., at Jerome, Ariz., advantage has been taken of this fact by providing a means of by-passing the water at the end of the water line into the air line in case an emergency arises.

Improving Working Conditions in a Hot Mine

By Russell C. Fleming, Miami, Ariz.

(Min. & Met., February, 95. 1700 words)

Methods used to increase ventilation in the mine of the Magna Copper Co. at Superior, Ariz., where there are high rock temperatures, hot water and high relative humidity.

Adjustable Pneumatic Brattice for Controlling Ventilation

By V. T. Berner, Ray, Ariz.

(Min. & Met., February, 97. 1800 words)

This apparatus was designed primarily to meet the demand for a quick, efficient stopping to seal off the burning area temporarily during a mine fire, but it can be used whenever an immediate brattice is required for ventilation control. It eliminates the need for using axes, hammers and saws while working in breathing apparatus. It can be erected by an ordinary miner, not familiar with rescue work.

Use and Cost of Compressed Air

By Robert L. Lewis, Salt Lake City, Utah

(Tech. Pub. No. 287; Class A, Metal Mining, No. 33; Class F, Coal and Coke, No. 31)

Theoretically compressed air is not utilized efficiently in the rock drill and yet no serious competitor to the air-driven drill has appeared. Maintenance of the compressed-air system in the best condition is necessary to attain reasonable approach to the high efficiency. Leaks in distributing lines must be avoided; the intake of the compressor should be from a supply of cool air. The choice between single-stage and two-stage compression depends largely on cost of power, altitude of the plant, capacity required, and the probable life of the plant. For capacities up to about 300 cu. ft. per minute a single-stage machine may be used for a pressure up to 100 lb., but for larger capacities and high altitudes the advantage lies distinctly with a two-stage compressor. The article concludes with a compilation of "plant" costs for sixteen different installations with a brief description of operating conditions.

Improving the Factor of Economy in Mine Ropes

By H. S. Cooley, Philadelphia, Pa.

(Min. & Met., May, 263. 3000 words)

The author concludes that the factor of economy in mine ropes, which is the ratio of number of trips made to the factor of safety, or the measure of economic efficiency in wire-rope usage, may be improved by: (1) a moderate increase in factor of safety as compared with average practice; (2) adoption of Lang lay; (3) punctilious maintenance of grooves; (4) use of sufficiently hard material for sheaves and drums.

Storage-Battery Locomotives

By Russell C. Fleming, Globe, Ariz.

(Min. & Met., November, 535. 3000 words)

This article presents the case of the storage-battery locomotives, pointing out that their use has increased tremendously in recent years as the knowledge of their proper application has spread. Within their limitations, lowered costs, increased efficiency and a lower accident rate result. Illustrations are given of typical mines where storage-battery locomotives are put to duty for which they are fitted and where they are giving satisfaction. They operate to best advantage on short hauls, with fairly intermittent service. Dissatisfaction in the past came from placing motors on duty for which they were underpowered.

How Human Beings Respond to Changing Atmospheric Conditions

By W. J. McConnell, New York, N. Y.

(Tech. Pub. No. 319; Class A, Metal Mining, No. 35; Class F, Coal and Coke, No. 33)

Human subjects were used for a series of experiments conducted in a two-compartment chamber, insulated by cork board, each compartment designed to maintain air conditions automatically at a desired temperature, humidity and air velocity. The air-conditioning apparatus consisted of fans, heaters, humidifiers, refrigeration equipment, distributing system and automatic control capable of maintaining any dry-bulb temperature from 20 to 180 deg. Fahr.; any humidity from 10 to 100 per cent; and a velocity from still air to 1000 ft. per minute. The paper records the results of the experiments as reflected in the effects of varying temperature and humidities on the human body.

Gases Which Occur in Metal Mines

By D. Harrington and E. H. Denny, Washington, D. C.

(Preprint; Class A, Metal Mining. 8300 words)

While it is true that methane occurs, or is likely to occur, in almost any or all coal mines, it is also a fact that the metal mines of the world have in them by far a greater variety of gases than have the coal mines. In metal mines there are a number of instances of occurrence of methane and attendant explosions; some metal mines have appreciable percentages of the decidedly explosive hydrogen; in some metal-mining regions carbon dioxide flows into the mine, fills the workings and overflows like water. In other regions, at many working faces there are high-temperature gases containing a mixture of gases of sulfur (sulfur dioxide, hydrogen sulfide and possibly SO_3 in some cases) together with carbon dioxide and nitrogen, and the deadly hydrogen sulfide in lethal proportions is found in some high-sulfide metal mines at blasting time. Carbon monoxide constitutes a decidedly dangerous day by day hazard in connection with metal-mine blasting, because of the heavy charges of dynamite that are

used and the use of fuse, which in burning gives off much carbon monoxide. The poisonous-gas hazard is accentuated by the relatively small openings where metal-mine blasting is done and by the inefficient ventilation of the usual metal mine.

Mining Methods and Systems

By THOMAS T. READ, New York, N. Y.
(Min. & Met., July, 336. 2400 words)

DEMONSTRATION of the fact that the terminology of mining is an unnecessary handicap to students in mining schools, and an appeal for suggestions tending toward clarification.

The Mining Methods Committee Reports

By F. W. BRADLEY, San Francisco, Cal.
(Min. & Met., January, 29. 3500 words)

A PLAN is described for cooperation between the U. S. Bureau of Mines and the A. I. M. E. in publishing papers on mining methods. Mechanization is discussed, including a description of a light machine used at Rico. Special problems are described, also innovations, such as the use of a drag scraper instead of mine cars on secondary levels. A brief bibliography is included.

Electric Motors in the Tri-State Field

By ROY BERENTZ, Picher, Okla.
(Min. & Met., June, 297. 2000 words)

PRACTICAL discussion of the factors involved in the use of electric motors in mining and milling operations.

Vertical Transportation in the Cœur d'Alene

By A. C. STEVENSON, Wallace, Idaho
(Min. & Met., May, 243. 4000 words)

DISCUSSION of all the factors that must be taken into account in designing a hoisting system for future as well as present needs.

Supply Trucks at the Copper Queen

(Min. & Met., August, 387. 2100 words)

DESCRIPTION of a truck that can be lowered through the supply shaft with its contents firmly wedged in place, which avoids the loading and unloading usually necessary in lowering supplies into a mine.

Visiting the Ashio Copper Mine

By S. L. GILLAN, Los Angeles, Cal.
(Min. & Met., April, 220. 1600 words)

ASHIO has been worked since 1620 and still produces 28,000,000 lb. of copper annually from a peculiar type of orebody.

Federal Mining Act of 1872 and the Problems of Its Amendment

By Archibald Douglas, New York, N. Y.
(Min. & Met., February, 79. 2500 words)

This paper quotes some sections of the law and suggests that a comprehensive survey be made so that if it ever becomes necessary to amend the law the necessary information will be available. The intricate web of federal and state mining laws and judicial decisions thereunder is not easy to unwind or to modify. Discussion in April, 225.

MILLING AND CONCENTRATION

Milling Methods in 1929

By Galen H. Clevenger, Boston, Mass.
(Min. & Met., January, 41. 1700 words)

This paper indicates that real progress has taken place, though few new devices or methods have appeared. Some problems have been solved and newer developments have been more firmly established.

Economic Points in Milling

By E. H. Crabtree, Jr., Baxter Springs, Kans.
(Min. & Met., August, 394. 1600 words)

This paper considers the economic points of milling for mills with different milling equipment, character and quality of feed, market conditions, and other variable factors.

Classifier Efficiency, an Experimental Study

By A. W. Fahrenwald, Moscow, Idaho
(Tech. Pub. No. 275; Class B, Milling and Concentration, No. 27)

The ratio "weight divided by surface" of sand grains is a widely varying factor for ore grains passing a given sieve aperture. Closed circuit classifier efficiency is not theoretically accurately expressed on the basis of sieve analysis. Closed circuit classifier efficiency shows to much better advantage on the basis of ideal classification than on the basis of sieve analyses. The efficiency of the classification studied is about 60 per cent. This suggests opportunity for useful further research in the field of this type of classification. Classifier efficiency on the basis of removal of finished product shows up to better advantage than on the basis of over-all efficiency.

Importance of Classification in Fine Grinding

By J. V. N. Dorr, New York, N. Y., and A. D. Marriott, Denver, Colo.
(Trans., Milling Methods, 109. 17,000 words)

Evolution of present fine-grinding flow sheets and varied use therein of classifiers is reviewed. Fine-grinding and classification practice of

nine Western copper concentrators are discussed in detail with flow sheet diagrams representative of layouts at various stages of development. Effectiveness of flow sheets is compared on the basis of rated section tonnage, unit power consumption, comparative screen analyses and metallurgical results subsequently secured. An ideal flow sheet is suggested, including preliminary desliming classifiers, primary closed-circuited mills, intermediate open-circuit classifiers, secondary closed-circuited mills and tertiary closed-circuited mills. There are addenda on selective classification results in South Africa.

A Laboratory Investigation of Ball Milling

By A. M. Gow, A. B. Campbell, and W. H. Coghill, Rolla, Mo.

(Tech. Pub. No. 326; Class B, Milling and Concentration, No. 29)

A NEW theory of ball action is advanced from a study of a 3-ft. squirrel-cage mill and a new formula of ball paths is derived. Laboratory tests with short ball mills in which slippage was reduced to a minimum show that the best grinding results were obtained at lower speeds than those hypothecated by previous theories. A speed of 65 per cent of the critical gave the maximum grinding, while a speed of only 50 per cent of the critical gave the most efficient grinding. By comparing the grinding results of the mills of various diameters, it was found that at the same per cent of the critical speed: *a*. The units of surface per unit weight varied as the 0.6 power of the diameter; *b*. The surface tons, or grinding capacity, varied as the 2.6 power of the diameter; *c*. The horsepower also varied as the 2.6 power of the diameter; *d*. The surface tons per horsepower-hour, or efficiency of grinding, was constant regardless of the diameter; and *e*. The units of surface per unit weight varied approximately as the peripheral speed. The larger mills showed larger grinding capacity per unit volume, but no increase in grinding efficiency.

Milling Practice at San Francisco Mines of Mexico, Ltd.

By Glenn L. Allen, Laredo, Texas

(Tech. Pub. No. 371; Class B, Milling and Concentration, No. 31)

THIS paper briefly mentions early attempts to treat San Francisco ore, then traces the development of concentration and flotation from 1903 until the present time. After giving the characteristics of the ore, the author describes the present milling practice consisting of gravity concentration to produce a small amount of lead concentrate, followed by flotation treatment for lead and then for zinc recovery. Important steps in the process are described in some detail. Screen analyses, reagents, assays and distributions of metals in the products, power consumptions, and other milling data are given.

Milling Methods and Costs at Presidio Mine of the American Metal Company of Texas

By VAN DYNE HOWBERT, New York, N. Y., and FRED E. GRAY, Moscow, Russia

(Tech. Pub. No. 368; Class B, Milling and Concentration, No. 30)

THIS paper supplements one recently published which described mining methods and costs at this Texas silver mine. The ore is siliceous, largely oxidized and contains, in addition to the silver values, several per cent lead. The mill is a cyanidation and gravity concentration plant of around 200 tons daily capacity, whose equipment includes Diesel engine power plant, gyratory crusher, Symons cone crusher, Hum-mer screen, tube mills, Dorr classifiers, Wilfley tables, Pachuca tanks, Dorr thickeners, Crowe-Merrill zinc dust precipitation plant, and Oliver filters. The details of the treatment process are given, including remarks on the use and consumption of chemicals, methods of conveying and sampling, and water supply. It is also stated that experimental work with flotation has indicated that its use would not give as good an economic return as does the present process. Tables give comparisons of operating results since 1883, screen analyses, consumption of various mill supplies, assays, recoveries, direct costs, efficiencies and other data.

Determination of Oxides in Ores and Mill Products

By DANIEL E. HUFFMAN, South Gate, Cal.

(Min. & Met., December, 591. 900 words)

METHODS for determining oxides in copper in connection with leaching operations and in ores and mill samples; oxides in iron pyrite, and in lead and zinc.

Concentration of Oxidized Lead Ores at San Diego Mill, Cia. Minera Asarco

By AUGUSTUS J. MONKS and NORMAN L. WEISS, Santa Barbara, Chih., Mexico

(Min. & Met., October, 455. 4000 words)

THIS paper describes the method employed in treating the oxidized lead ores of Santa Barbara, Chihuahua, Mexico. The natural difficulties of the process are augmented by the complexity and variety of the ores and ore minerals. The ore, after being ground in rod mills, is deslimed and tabled for a coarse lead concentrate. The table tailing is reground and combined with the thickened slimes to compose the flotation feed. Successful flotation depends upon the correct use of sodium silicate, sodium sulfide, and "San Diego Mixture." The last is a heavy oil formula especially developed for this treatment. A detailed description of these reagents and the method of using is given. The San Diego mill has been treating 600 to 700 tons of oxidized lead ores per day since 1921.

United Verde Milk of Lime Valve

(Min. & Met., January, 50. 500 words)

DESCRIPTION of an apparatus developed by L. M. Barker for feeding of lime in the mill of the United Verde Copper Co., at Clarksdale, Ariz., that is easily and cheaply operated, free from dust and is remarkable in the simplicity and ease with which the amount of lime fed is controlled.

Chemical Reactions in Flotation

By ARTHUR F. TAGGART, T. C. TAYLOR and A. F. KNOLL, New York, N. Y.

(Tech. Pub. No. 312; Class B, Milling and Concentration, No. 28)

IT IS postulated that, in flotation with soluble reagents, all phenomena governing collection are controlled through simple chemical reactions between the reagents and compounds occurring at the surfaces of the particles affected. Evidence, thought to be conclusive, is offered in support of this hypothesis. Considerable detailed experimental evidence of the nature of the chemical reactions between particle surfaces and flotation reagents is given.

When certain finely divided substances are suspended in certain fluid media, and the suspension is examined under sufficiently high magnification to resolve the smaller particles clearly, the mass of fine particles is seen to be in erratic motion, which, for any particle that is moving, is continuous, and which continues more or less indefinitely, so long as surrounding conditions remain reasonably constant. This phenomenon is known as the Brownian movement.

Evidence is presented tending to disprove the classical explanation of Brownian movement. An alternative hypothesis of the mechanism of Brownian movement is set forth, and evidence in support thereof adduced. Complete correlation between Brownian movement and the collection phenomena in differential flotation of lead-zinc-iron-silica is shown. The use of Brownian movement and the new hypothesis as a powerful tool in the investigation of flotation phenomena is demonstrated.

Effect of Xanthates, Copper Sulfate and Cyanides on Flotation of Sphalerite

By A. M. GAUDIN, Butte, Mont.

(Trans., Milling Methods, 417. 3600 words)

FROM a study of the effect of xanthates, copper sulfate and cyanides on the flotation of sphalerite the following conclusions appear justified: (1) Pure sphalerite does not abstract ethyl xanthate from solution and is not floated thereby. (2) Cyanide solutions form no coating on sphalerite but lead to the solution of some zinc probably as a complex zinc-cyanide ion. Pure sphalerite is not depressed by cyanide. (3) Sphalerite readily acquires from copper-bearing solutions a coating of covellite having a thick-

ness of a few atomic diameters; thereafter the reaction is slow. (4) Covellite-plated sphalerite abstracts xanthate from solution and is floated thereby. (5) The covellite coating of sphalerite previously treated with copper sulfate is readily dissolved by cyanide solutions. Cyanide solutions therefore have a cleansing effect on covellite-plated sphalerite surfaces. (6) Cyanide solutions greatly decrease the floatability of copper-activated sphalerite, bringing it back to the normal floatability of pure sphalerite. (7) Natural sulfide zinc ores contain sphalerite in various stages of activation by copper salts and therefore call for the use of various amounts of cyanide and of copper sulfate in selective flotation.

Chemical Tools of Flotation

By G. H. Buchanan, New York, N. Y.

(Min. & Met., December, 565. 6000 words)

In order to make the nomenclature of the chemical tools of flotation more real, the writer likens flotation to an old-fashioned balloon ascension. The mineral to be floated is the aeronaut, the small boys standing about are the gangue, the oil film surrounding the bubble is the counterpart of the silken fabric of the balloon, and bubble and balloon will rise for the same reason; namely, that the fluid outside it is heavier than the fluid within. By way of illustration the comparison is maintained throughout the paper. The chemical "collectors" are the counterpart of the rigging of the balloon, and so on. The writer thus discusses activators and dispersing agents, frothers, promoters, depressors, regulators, etc.

Selectivity Index; a Yardstick of the Segregation Accomplished by Concentrating Operations

By A. M. Gaudin, Butte, Mont.

(Trans., Milling Methods, 483. 2000 words)

Selectivity index is a *single-number* quantitative measure of the segregation effect between metals (or minerals) as the result of concentrating operations. It is designed for use as an adjunct to the usual metallurgical criteria.

If x is the recovery of substance A, and y the rejection of B, the selectivity index is

$$I = \sqrt{\frac{x}{100-x} \cdot \frac{y}{100-y}}.$$

Also if a and a' are the A content of the concentrate and tailing, and b and b' are the B content of the concentrate and tailing, the index is

$$I = \sqrt{\frac{a}{a'} \cdot \frac{b'}{b}}$$

Examples of the application of this index are given.

IRON AND STEEL

The Future of the American Iron and Steel Industry

By Zay Jeffries, Cleveland, Ohio
(Tech. Pub. No. 331; Class C, Iron and Steel, No. 54)

Although many of the specific changes that will come in the future cannot be foreseen, it can be said there are no factors outside the iron and steel industry in sight which seriously threaten its future growth. The industry, however, needs more research and development work, both to pay for sins of omission in the past and to provide aggressive but intelligent exploitation of iron and its various and wonderful alloys for the betterment of mankind. Assuming reasonable compliance with the various conditions discussed, it would appear that the American iron and steel industry can look forward toward not only a healthy but even a romantic future.

Sintering Limonitic Iron Ores at Ironton, Minnesota

By Perry G. Harrison, Ironton, Minn.
(Tech. Pub. No. 284; Class C, Iron and Steel, No. 46)

This paper describes in detail the first application of sintering to limonitic iron ores on a Dwight-Lloyd machine, reducing the moisture about 15 per cent and improving its physical structure. The flow-sheet of the plant, analyses of raw materials and finished products and details of costs are given. The author concludes with a general discussion of the economic advantages of sintering, and points out that if the blast furnaces would pay for the advantages which the use of sintered ore affords them the cost of the sintering operation could be met by producers.

Experiments Demonstrate Method of Producing Artificial Manganese Ore

By T. L. Joseph, E. P. Barrett and C. E. Wood, Minneapolis, Minn.
(Tech. Pub. No. 310; Class C, Iron and Steel, No. 51)

Tests conducted with a 6-ton blast furnace indicate that a charge of 100 per cent manganiferous iron ore can be melted successfully. Such an ore charge would produce high-phosphorus spiegel containing 10 to 15 per cent manganese and about 0.5 per cent phosphorus. The manganese can be separated from the phosphorus and iron in this type of metal by treating it in a standard basic open-hearth furnace or an electric furnace. By adding iron ore most of the manganese, together with some of the phosphorus, is oxidized and passes into the slag. At this point the slag is composed largely of FeO and MnO. The FeO and the phosphorus compounds can be reduced much more rapidly than the MnO by covering the slag with a layer of coke. By holding the slag under a layer of coke for about 3 hours in the open hearth and 1 hour in the electric furnace, the FeO and phosphorus can be reduced to amounts

permissible in ferro-grade materials. As the FeO is reduced, low-silica slags become very viscous, but this can be overcome by adding silica or regulating the silica in the iron ore. Silica in the slag is objectionable, but no other effective cheap thinning material has been found. Changes in the composition of the slag and metal of open-hearth and electric-furnace heats are shown graphically. These experiments are significant because they indicate that large quantities of ferromanganese can be made from Minnesota manganiferous iron ores. The method can be turned to quickly, and a large part of our ferro requirements can be met without serious loss of equipment needed for the production of steel. Discussion, May, 278.

Resistance of Iron Ores to Decrepitation and Mechanical Work

By T. L. Joseph and E. P. Barrett, Minneapolis, Minn.

(Tech. Pub. No. 372; Class C, Iron and Steel, No. 58)

Excessive amounts of fine ore are detrimental to good blast-furnace practice. Although the fine particles expose a large surface area, they tend toward high pressures, slow rates of blowing, irregular stock travel, poor gas distribution and large dust losses. Methods of filling fine ores, which permit regular stock descent at fast driving rates, do so by virtue of poor gas distribution; some portion of the furnace acts as a relief valve through which the gas may pass without building up pressure that interferes with regular stock descent.

Surveys of conditions within the stock column of several industrial furnaces show that gas distribution is far from ideal. Methods of filling and furnace lines are important, but the improved gas distribution in furnaces operating on coarser ores and sized ore indicates that some combination of crushing, sizing and sintering would offer the most complete solution to good gas distribution.

Thirty-four samples of ore, including a variety of iron ores and several manganese ores, were tested by an arbitrary procedure to determine their relative resistance to cold work and to hot work. The results show that screen analysis alone does not give complete information on ore structure. A more detailed study should be made to determine the extent of crushing or the most logical ores to sinter. In extreme cases the average particle size was only 15 to 35 per cent as large after the ore had been subjected to mild mechanical treatment at temperatures up to 1400° F. Discussion is in Transactions (1930).

Concentration of the Mesabi Hematites

By E. W. Davis, Minneapolis, Minn.

(Min. & Met., November, 518; Trans., Iron and Steel Div., 358. 2700 words)

The Mesabi Range ships annually about 35,000,000 tons of iron ore assaying 8 per cent in silica, but this tonnage and grade can be main-

tained only because ores assaying 4 or 5 per cent silica can be secured from some of the mines to mix with ores from other mines assaying 10 to 15 per cent silica. When the high-grade ores are gone, the low-grade ores cannot be mined without materially reducing the grade of the shipping products. The remedy for this is the beneficiation of the low-grade ores. Although one-third of the total ore shipped from the Mesabi Range each year is beneficiated, only a fraction of this is so treated as to reduce the silica content. Washing is by far the most important of the silica-removing processes, but the true wash ore on the Mesabi Range is being used up rapidly. The enormous tonnage of low-grade ore remaining in the range requires more elaborate methods of treatment for the production of low-silica, high-iron concentrate. Jigging is receiving careful attention and undoubtedly will be developed further. Table concentration, once used extensively, will probably come back into use. Magnetic concentration is being considered and seems to offer a possible method for treating nearly any of the low-grade ores. No commercial installations or roasting furnaces for the conversion of hematite to magnetite have been made and until full-scale equipment has been tested, operating costs will not be known. Estimates place the cost of magnetic roasting at from 40c. to 50c. per ton and the cost of crushing, magnetic separation, and sintering a part of the product will bring the total cost of treatment to about $1.50 per ton of final product. With the present price of iron ore, the economic value of this method of concentration is very doubtful but when ore prices increase, as they undoubtedly will, this method of treatment can be used to produce low-silica concentrate from nearly all of the low-grade ore materials at a cost that will be attractive.

Beneficiation of Iron Ores from the Blast-furnace Viewpoint

By Ralph H. Sweetser, Columbus, Ohio

(Min. & Met., September, 423. 4700 words)

Résumé of past and present practice in beneficiation, citing opinions of various writers and of Dean Appleby, of the School of Mines of the University of Minnesota. The paper and the following discussion by T. T. Read indicate that not only more data, but more accurate data, are needed to help in the solution of the problems involved. (Discussion in Transactions (1930).

Some Aspects of the Iron Ore Situation

By F. B. Richards, Cleveland, Ohio

(Min. & Met., September, 437. 2300 words)

This paper reviews the subject of iron-ore reserves, necessity of beneficiation and distribution, especially of Lake Superior ores. Discussion in Transactions (1930).

Iron and Steel Metallurgy in 1929

By G. B. Waterhouse, Cambridge, Mass.
(Min. & Met., January, 34. 3300 words)

Records progress and activity in the iron and steel industry in the United States. New processes were developed and improvements made in coke-oven and blast-furnace practice and in steelmaking. A program for research on alloy steels was initiated. Improvements were made in cast iron and wrought iron.

Columbia Steel Corporation Operations

By W. R. Phibbs, Salt Lake City, Utah
(Min. & Met., April, 205. 2000 words)

This paper describes the operation of one blast furnace at Ironton, Utah, in which screening the ore and charging it into the furnace in layers of the same size has resulted in a marked decrease in coke consumption. It also gives a brief description of a tin mill in California.

Problems of Steel Plant Metallurgy

By Wilfred Sykes, Chicago, Ill.
(Min. & Met., May, 256. 5200 words)

The author sums up various problems in coke-oven, blast-furnace and open-hearth work and concludes that the first need is to determine more fundamental data in regard to the open-hearth process and the second is greater cooperation between plants in the application of such data.

Charcoal Blast-furnace Practice in Mysore

By B. Viswanath
(Min. & Met., July, 332. 4000 words)

Details of the construction and operation of a blast furnace that uses a low-silica ore with a high-alumina slag.

Sponge Iron and Its Relation to the Steel Industry

By Edward P. Barrett, Minneapolis, Minn.
(Min. & Met., August, 395. 1500 words)

The author believes that high-grade sponge iron can be made only from high-grade raw materials, that it must be melted in the electric furnace, that it should be considered only as a raw material for the production of steel, and that no sponge-iron process can be a competitor of the blast furnace for the production of pig iron.

Electrolytic Iron from Sulfide Ores

By Robert D. Pike, George H. West, L. V. Steck, Ross Cummings and B. P. Little, Emeryville, Cal.
(Tech. Pub. No. 268; Class C, Iron and Steel, No. 41)

This paper describes in detail a pilot-plant operation for producing iron of almost complete chemical purity and in dense homogeneous

form from sulfide ores or materials containing metallic iron. This process requires the use of a cell with diaphragms and a hot electrolyte. When the sulfide ores contain other metals, these are recovered as byproducts, and even when copper concentrates are treated it is necessary to look upon iron as the main product and upon the copper as a secondary product. An outline is also given of a process for producing pure electrolytic iron as porous cathodes from scrap, which employs a simple non-diaphragm cell and a cold electrolyte. Either of these processes is capable of rendering available, on a large scale and at a reasonable cost, iron of almost complete chemical purity, and their commercial development awaits only a greater demand for pure iron. Discussion, April, 212.

The Permanent Growth of Gray Cast Iron

By Walter E. Remmers, Chicago, Ill.

(Tech. Pub. No. 337; Class C, Iron and Steel, No. 55)

The phenomenon of irreversible growth of gray cast iron can be described as a result of: 1. Precipitation, solution and reprecipitation of graphite in the solid matrix material. Ordinarily this effect is predominant in the growth obtained on the first heating after casting. 2. Oxidation of the matrix material after the graphite flakes have been burned out. This reaction has a large accelerating effect on growth. 3. Mechanical swelling created by finely fracturing the slightly ductile structure of gray cast iron. This fracturing is most effective on passing through the A_1 transformation both on heating and cooling. All alloying elements, such as silicon, aluminum and under certain conditions titanium and nickel, which favor the precipitation of carbon, tend to increase growth while elements such as chromium and manganese, which exert a stabilizing effect on the carbon, tend to decrease growth. Increase in total carbon increases the growth. Fineness in the dissemination of graphite is conducive to a retarded growth. Discussion, August, 379.

Production of Gray Iron from Steel Scrap in the Electric Furnace

By T. F. Baily, Canton, Ohio

(Tech. Pub. No. 296; Class C, Iron and Steel, No. 48)

This is a description of the first successful attempts (Canton, Ohio, 1927) at producing gray iron with silicon and manganese additions reduced directly in the furnace. The furnace used is first described and illustrated by a photograph, the procedure is described and tables of detailed analyses of the raw materials and product are given. It had a high pearlitic content, usually 10 per cent above cupola iron. The Brinell hardness was slightly less while the chemical composition and machining qualities were at least equal to those of blast-furnace iron, while the specific gravity was higher. It showed a higher deflection than blast-furnace iron of the same composition; one of its outstanding characteristics was that regardless of the temperature of the

molds there was very little evidence of chill in the iron. The author concludes that where scrap steel can be obtained at a relatively low price electric pig iron can be made economically, with qualities superior to pig iron produced in the cupola.

Development of Casing for Deep Wells

By F. W. BREMMER, Ambridge, Pa.

(Tech. Pub. No. 355; Class C, Iron and Steel, No. 57; G, Petroleum, No. 34)

THE production of oil from the deeper wells of today requires a casing material which is capable of resisting high stresses. The requirements for satisfactory deep-well casing are briefly discussed. Six types of steels were studied to determine their suitability for service in casing deep wells. This study indicated that one type, silicon manganese chrome, offered a solution to the problem. The characteristics of commercial casing made from this SiMnCr steel were thoroughly studied. Tables of physical properties revealing the desirable combination of high strength with good ductility and toughness are given for the SiMnCr type casing. The development of a flattening test, performed on every piece of casing produced to insure satisfactory ductility, is also described.

Endurance Properties of Steel in Steam

By T. S. FULLER, Schenectady, N. Y.

(Tech. Pub. No. 294; Class C, Iron and Steel, No. 47)

THE endurance limit of nickel steel, both A and B specimens, is shown to be lower in 60-lb. steam at 150 deg. to 160 deg. C. than in air at room temperature. The endurance limit of nickel steel B specimens, in the presence of steam, condensed moisture and an excess of oxygen has been shown to be not more than 36.5 per cent of the endurance limit of similar specimens tested in the atmosphere at room temperature, and in steam containing 0.208 per cent oxygen at 98 deg. C. An endurance limit of 65,000 lb. per sq. in. was obtained in 60-lb. steam at 150 deg. to 160 deg. C. with specimens of a nitrided steel.

Alloy Steels

By C. E. MACQUIGG, Long Island City, N. Y.

(Min. & Met., December, 578. 2300 words)

TRACES the development of commercial use of alloy steels from the issuing of one of the earliest patents, in the sixties, which covered the use of chromium. Tungsten, heat-treating to improve cutting speeds, nickel steels and stainless steels followed in succession. The first use was for tools, then for ordance, machine parts and engineering purposes. About 1920, novel metallurgical achievements began. For purposes where the brittleness of high-alloy steels was detrimental, medium alloys were developed. Manganese steel is now used for bridge con-

struction, for rails and for cylinders to contain gases at high pressures. Nickel, manganese and chromium steels are widely used, and continued development is to be expected.

Practical Observations on Manufacture of Basic Open-Hearth, High-Carbon Killed Steel

By W. J. Reagan, Oakmont, Pa.

(Tech. Pub. No. 347; Class C, Iron and Steel, No. 53)

The material described is bottom-cast basic open-hearth steel of 0.5 to 0.85 per cent carbon, phosphorus and sulfur 0.04 per cent maximum, silicon 0.15 to 0.35 per cent and manganese 0.5 to 0.75 per cent. It is cast in 12-sided ingots of standard length. The paper represents the results of several years' study of the defects in basic open-hearth steel, and of the results on such defects obtained by various changes in practice. Raw materials are first taken up, next the design of the ingot mold, ladles, practice in the furnace, pouring practice and the effect of analysis in rejections. He concludes with a claim that as good forging steel can be made in the basic open hearth as in the acid furnace, the phosphorus and sulfur often being lower than is common in acid open-hearth steel.

Rate of Carbon Elimination and Degree of Oxidation of Metal Bath in Basic Open-hearth Practice—II

By A. L. Feild, Lockport, N. Y.

(Tech. Pub. No. 280; Class C, Iron and Steel, No. 45)

Overoxidation of the metal bath during refining by the basic open-hearth process is an inherent feature. It may be reduced to a minimum only by obtaining a relatively slow rate of carbon drop and by the maintenance of a slag as low as practicable in effective FeO content. Sufficient data are not available to determine whether or not a heat is in better condition with respect to FeO during a very slow boil than it is after it has come to practical equilibrium under the "flat" slag which characterizes completion of the boil, carbon content, FeO content of slag, and temperature being the same in both cases.

A New Method for Determining Iron Oxide in Liquid Steel

By C. H. Herty, Jr., J. M. Gaines, Jr., H. Freeman and M. W. Lightner, Pittsburgh, Pa.

(Tech. Pub. No. 311; Class C, Iron and Steel, No. 52)

A method of determining the iron oxide content of liquid steel is described. The method consists of adding aluminum to steel in a test-spoon, pouring the killed steel into a suitable mold, and analyzing the metal for Al_2O_3, from which the FeO in the steel may be calculated. From the experiments carried out to date, the method gives every indication of being quantitative. Results on a basic and an acid open-hearth heat have been included in this paper, but no attempt has been made

to go into the theory of steel refining on the basis of these two heats, as the data will be used in conjunction with other heats for this purpose. Some outstanding features, however, have been indicated.

The Transformation of Austenite at Constant Subcritical Temperatures

By E. S. Davenport and E. C. Bain, Newark, N. J.

(Tech. Pub. No. 348; Class C, Iron and Steel, No. 56)

This paper sets forth certain characteristics of the austenite transformation in steels first heated to form a complete austenite solid solution and subsequently cooled rapidly to various subcritical temperatures permitting transformation. The characteristics investigated are: 1. The velocities of transformation at various constant temperatures. 2. The hardness of the products of transformation resulting at various constant temperatures. 3. The structures resulting from the transformation being forced to occur at various constant temperatures. 4. The influence of carbon (and other elements) on the characteristics mentioned above. In general it has been found that transformation is slow immediately below the eutectoid temperature, very rapid a few hundred degrees lower and very slow for most ordinary steels between 300° F. and 550° F. (150° C. and 300° C.). The reaction is again rapid at still lower temperatures. High-alloy steels show the same characteristic velocity changes at somewhat different temperatures. The great variation in time involved in the transformation at various temperatures necessitated the use of a logarithmic time scale in plotting the duration of the reactions. The hardness of the product decreases with increase in the temperatures at which the transformation occurs. The hardness varies, not in a linear fashion, but in a characteristic manner. The structures are arbitrarily designated by five typical microscopic appearances. The names applied to the structures are in no wise to be regarded as significant of anything novel other than convenience. The authors have named them, A. Coarse Pearlite, B. Fine Pearlite (often classed as nodular troostite), C. Troostite, D. Martensite-troostite and E. Martensite.

Tensile Properties of Rail and Other Steels at Elevated Temperatures

By John R. Freeman, Jr. and G. Willard Quick, Washington, D. C.

(Tech. Pub. No. 269; Class C, Iron and Steel, No. 42)

This study made in the temperature range 400° to 700° C. showed that the ductility of all rail steels decreases with increase in temperature in portions of the range, becoming less between 500° and 650° C. than at ordinary temperature. This is called the "secondary brittle" range to distinguish it from the other two ranges known. The results indicate that it is a property of a heat as a whole. The cause has not been determined, though some evidence indicates it may be related to

the MnO content or the free carbides. A hypothesis as to the cause of shatter cracks, based on this phenomenon is advanced, and it is suggested that unexplained failures in other types of steels may be due to this cause. The results indicate that slow cooling through the secondary brittle range is desirable to avoid shatter cracks.

Influence of Rate of Cooling on Dendritic Structure and Microstructure of Some Hypoeutectoid Steel

By A. Sauveur, Cambridge, Mass., and C. H. Chou, China

(Tech. Pub. No. 299; Class C, Iron and Steel Division, No. 49)

From the results reported in this paper and from those of many other experiments, the following conclusions appeared justified. The dendrites of the commercial steel are considerably larger and more clearly defined than those of the pure steel. For like treatment the macro-grains of the commercial steel are smaller than those of the pure steel. For like treatment the Widmanstätten structure after slow cooling through the thermal critical range is much more pronounced in the pure steel. The dendrites of the four samples of commercial steel slowly solidified are substantially larger than those of the four samples more rapidly solidified. In the pure steel, there is no appreciable difference between the dendritic pattern of the samples slowly and rapidly solidified. The slower the cooling through the granulation range, the larger the macro-grains. The larger the macro-grains, the more marked the Widmanstätten type of structure after slow cooling through the thermal critical range. The less pronounced the dendritic structure, the more pronounced the Widmanstätten structure after slow cooling through the thermal critical range. Slow cooling through the thermal critical range induces the formation of a Widmanstätten structure; slow cooling induces the formation of a network structure. While these conclusions refer only to the two steels investigated, it is believed that they would be found applicable to carbon steels in general.

Progress Notes on the Iron-silicon Equilibrium Diagram

By Bradley Stoughton and Earl S. Greiner, Bethlehem, Pa.

(Tech. Pub. No. 309; Class C, Iron and Steel, No. 50)

The existing data on the constitution of the iron-silicon alloy were compiled and correlated, and an investigation was carried out on iron-silicon alloys containing less than 10 per cent silicon. This investigation showed that discontinuities occur in the temperature-electric resistance curves of the alloys studied, and these discontinuities correspond fairly well with the line of critical ductility in the iron-silicon equilibrium diagram as drawn by Pilling. Micrographic and hardness tests were made on the alloys. Finally a tentative equilibrium diagram of the iron-silicon system was proposed.

Production and Some Properties of Large Iron Crystals

By N. A. Ziegler, East Pittsburgh, Pa.

(Tech. Pub. No. 273; Class C, Iron and Steel, No. 43)

Large crystals of iron were prepared by the "Edwards & Pfeil" method and investigated. The preparation of large crystals from round bars is connected with difficulties, due to the non-uniformity of the stress distribution. Large crystals were prepared from flat strips of Armco iron and some of their mechanical properties investigated. Proportional limit, yield point and ultimate strength decrease with increase of grain size. Rings of vacuum-fused electrolytic iron were converted into large crystals by compression and annealing. Their magnetic permeability is several times higher than that of similar but poly-crystalline samples, confirming previous results to the effect that there is a definite relationship between magnetic properties and grain size.

NONFERROUS METALLURGY

The Leaching Process at Chuquicamata, Chile

By Charles W. Eichrodt, Laurel Hill, N. Y.

(Tech. Pub. No. 350; Class D, Non-ferrous Metallurgy, No. 28)

After describing the occurrence of the ore and its characteristics, from a metallurgical standpoint (of which the chlorine and nitrate content are of the most importance) the crushing operations are described followed by the leaching operations, which are in two stages; the dissolving of the copper from the ore and the washing. The dissolving operations yield a strong solution which is sent to the tankhouse for electrolytic precipitation. The washing consists of downward displacement of the strong solution by a series of four wash solutions. The tailing retains about 8¼ per cent moisture. The accumulation of deleterious elements is met by discarding spent electrolyte or post-treatment solution. The leaching cycle, 84 hours, is made up of 55 hr. of soaking, 16 hr. washing and drawing, 6½ hr. unloading, and the remainder delays. A series of charts gives an unusually clear interpretation of the leaching process and the paper describes in detail the leaching and dechloridizing plants and their equipment.

Progress in the Reduction and Refining of Copper, 1929

By Frederick Laist, New York, N. Y.

(Min. & Met., January, 33. 1200 words)

An intensive study of reverberatory-furnace smelting reveals no marked advantages from variations in methods of charging. Concrete is replacing wood as backing for copper electrolyzing cells. Potrerillos process employs solution purification in connection with electrolysis.

Important Steps in the Advance of Copper Metallurgy

By Eugene A. White, Tacoma, Wash.

(Min. & Met., January, 56. 2200 words)

A concise historical review of the various steps that have led up to the present high state of the metallurgical technology of copper.

Improvements in Copper Reverberatory Construction and Practice

By A. H. Richards, Tacoma, Wash.

(Min. & Met., June, 299. 1700 words)

Modern furnaces compared with early ones, indicating that the most radical improvement has been an increase in outlet area. Dimensions of furnaces from 1800 to 1918 are given.

Effect of High-grade Concentrates in Reverberatory Practice at Anaconda

By James L. Dougherty, Anaconda, Mont.

(Min. & Met., August, 389. 3450 words)

General discussion of developments in reverberatory smelting practice during the past five years at the plant of the Anaconda Copper Mining Company.

New Bins for Roaster Calcines at Douglas Smelter

By Russell C. Fleming

(Min. & Met., August, 397. 1800 words)

Detailed description and comparison of bins formerly and at present in use at the smelter of the Phelps Dodge Corporation.

El Paso Refinery of the Nichols Copper Company

By Frank R. Corwin and C. S. Harloff

(Min. & Met., October, 459. 7200 words)

The Nichols Copper Co., associated with the Phelps Dodge Corpn. and the Calumet and Arizona Mining Co., constructed during 1929 and is now operating a copper refinery at El Paso, Texas, with an annual capacity of 100,000 tons of anodes. These anodes are produced at the smelters of the companies and shipped by rail to the electrolytic refinery, where they are drill-sampled and then loaded into concrete lead-lined electrolytic tanks. The pure electrolytic copper produced is refined in 200-ton reverberatory furnaces using natural gas for fuel and equipped with waste-heat boilers and air preheaters. The refined copper is cast on Clark casting wheels, into various shapes and sizes suited to the industry. The silver slimes are treated in a specially designed Nichols-Herreshoff furnace, then they are either dried and shipped for refining or treated in a Doré furnace installation to make silver anodes.

Natural Gas Firing at El Paso Smelting Works

By E. R. Marble, El Paso, Texas

(Min. & Met., October, 466. 1400 words)

At the El Paso Smelter natural gas has replaced fuel oil entirely and no oil has been used for some months past. The change to gas throughout the plant was comparatively simple except at the copper reverberatory, where the necessity for high temperature demanded a careful study. Gas has proved satisfactory at the reverberatory without the necessity for carburization or preheating of the air. The flame is nonluminous but the necessary heat transfer is obtained through conduction and convection as well as by radiation. All problems have been successfully met with the gas installation.

A Petrographic Study of Lead and Copper Furnace Slags

By Roy D. McLellan, Maurer, N. J.

(Tech. Pub. No. 305; Class D, Non-ferrous Metallurgy, No. 25)

The ease with which a given furnace charge will smelt depends on the physical properties of the compounds produced in the furnace. The nature of these compounds can be learned from the study of the slags. The ultimate goal for studies in slags is to bring about the formation of desirable compounds and prevent or reduce to the minimum the formation of undesirable ones. The problems of lead and copper furnace slag are complicated by the difficulty in studying, under controlled conditions, silicate melts containing iron and sulfur as essential ingredients. This paper has resulted from the study of thin sections, from a large number of commercial lead and copper furnace slags, with the help of the petrographic microscope. It is an analytical survey of these slag problems designed to simplify the course to be followed in future slag studies by synthetic fusions.

Lead Refining at the Bunker Hill Smelter of the Bunker Hill & Sullivan Mining & Concentrating Co.

By A. F. Beasley, Kellogg, Idaho

(Tech. Pub. No. 303; Class D, Non-ferrous Metallurgy, No. 24)

Some five years ago this plant adopted the practice of making two crusts in the desilverizing kettles. After the usual dressing and softening of the bullion it contains 1.2 oz. gold and 100 oz. silver per ton and goes to the desilverizing kettles where first a gold crust, containing practically all the gold and ⅛ of the silver is obtained and then a silver crust, which contains ⅞ of the silver and a trace of gold. The gold crust is retorted, cupelled and parted with sulfuric acid in the ordinary way. The silver crust is liquated to remove the lead (48 per cent of the total weight) and then retorted to a bullion, containing 12,800 oz. of silver. This is cupelled to a fineness of 997.4 and melted in a Monarch furnace where it is brought to 999. Details as to the operations and handling by-products are given in the original paper.

Metallurgical Control at the Tooele Concentrator

By O. E. Keough, Tooele, Utah

(Min. & Met., April, 202. 3650 words)

Detailed description of routine tests by which technical control is maintained at the custom lead-zinc ore concentrator.

Investigation of Anodes for Production of Electrolytic Zinc

By H. R. Hanley, C. Y. Clayton and D. F. Walsh, Rolla, Mo.

(Tech. Pub. No. 321; Class D, Non-ferrous Metallurgy, No. 27)

Anodes used in the electrolysis of zinc sulfate solution are composed essentially of lead. However, impurities are always present and certain alloys have been tried. The purpose of this investigation is to ascertain the effects of various alloying metals on the resulting zinc and on power consumption. Silver in the anode lowers the electromotive force of the cell and decreases the quantity of lead in the cathode zinc. Calcium-lead anodes lower the anode potential but do not prevent the transfer of some lead to the cathode. If 1 per cent silver is introduced, in addition, the transfer of lead is prevented and at the same time a lowering of the potential is effected equivalent to 40 or 50 per cent of that applying to pure lead anodes.

Review of the Zinc Industry for 1929

By Frank W. Harris, East St. Louis, Ill.

(Min. & Met., January, 6. 1300 words)

This review indicates that developments of orebodies, successful concentration of ores hitherto impossible of utilization and increased reduction capacity have been the outstanding activities of the year.

Quicksilver Industry in 1929

Improvements in the Metallurgy of Quicksilver. By L. H. Duschak

Symposium on The Present Status of the Quicksilver Industry. Arranged by Charles G. Maier

With Discussion

(Tech. Pub. No. 264; Class A, Metal Mining, No. 29; Class D, Non-ferrous Metallurgy, No. 22)

The first paper shows that the direct furnace treatment of quicksilver ores remains the standard practice and when suitable attention is given to the design of the plant it leaves little to be desired. The problem of treating a large tonnage of ore below the necessary furnace grade may be solved under favorable conditions by wet screening followed by flotation and the retorting of the concentrate. There is a very apparent need for economical methods for use during the development of a property or where operations are necessarily on a small scale. Hand sorting and retorting have been the chief reliance in this connection. However, this practice is successful only where a relatively high-grade product, say 5 to 10 per cent Hg, can be economically

produced by sorting. When this is not possible a small furnace plant or a small flotation plant and battery of retorts offer the best solution.

The symposium arranged by Mr. Maier discusses the economic history of quicksilver, its geology, mining and metallurgy, and the relation of governmental agencies to the industry. It defies abstracting and should be read by all who are interested.

Progress in the Production and Use of Tantalum

By George W. Sears, Reno, Nev.

(Tech. Pub. No. 279; Class D, Non-ferrous Metallurgy, No. 23)

Tantalum is used in the manufacture of vacuum tubes, especially for grid wires, of electrodes, dishes, spatulas and other laboratory apparatus. More recently dental and surgical instruments, pen points, and hypodermic needles have been made of the metal because of its extreme resistance to corrosion. Various alloys containing tantalum have also found use for miscellaneous purposes. The metal must be regarded as a rare one, it being estimated that the crust of the earth contains more gold than tantalum. It occurs in combination with columbium and the extraction and separation is difficult. The development of more uses for columbium would effect some reduction in the cost of tantalum and stimulate the use of the metal.

Electrolytic Cadmium Plant of Anaconda Copper Mining Company at Great Falls, Mont.

By W. E. Mitchell, Great Falls, Mont.

(Tech. Pub. No. 320; Class D, Non-ferrous Metallurgy, No. 26)

At Great Falls the Anaconda Copper Mining Co. operates the largest cadmium plant in the world. The product, deposited by electrolysis, is 99.95 per cent pure. The cadmium is a by-product of the treatment of zinc concentrates. These are roasted and leached with sulfuric acid. After removal of the copper, cadmium is precipitated with zinc dust. The sponge is partly oxidized and then dissolved with sulfuric acid, the resulting liquor going to the electrolytic cells. An electric furnace is used for melting the cathodes with caustic soda. Marketable shapes include pencils, slabs, anodes and balls.

INSTITUTE OF METALS DIVISION

Recent Developments in the Melting and Annealing of Non-ferrous Metals

By R. M. Keeney, Hartford, Conn.

(Tech. Pub. No. 286; Class E, Institute of Metals, No. 98)

Recent developments in the melting and annealing of non-ferrous metals indicate that it is now generally understood to a greater extent than ever before that a comparison of costs of sources of heat on a

thermal basis means nothing without a complete investigation of overall costs, the only cost figure that results in profit or loss; that profit or loss does not necessarily result from individual process economy; and that the source of heat best suited to one operation may not fit another. These realizations are resulting in a definite trend toward the increasing use of the more highly refined sources of heat, electricity and gas, in non-ferrous metallurgy.

Melting and Casting Some Gold Alloys

By Edward A. Capillon, Attleboro, Mass.

(Tech. Pub. No. 282; Class E, Institute of Metals, No. 95)

Defects in gold and silver alloy ingots and their causes are described. It is shown that a correctly proportioned mold is necessary in order to obtain ingots to give as little scrap metal as possible. Gases in gold and silver alloys produce blisters when the alloy is rolled down and annealed. These gases can be rendered harmless by the use of deoxidizers. The characteristics of various deoxidizers are discussed. Insofar as ductility measured by the Olsen test is concerned casting temperature has apparently little effect on the ductility of rolled and annealed sheet. A summary is given of the effects of various impurities on fine gold. The harmful effect of lead in low-carat red golds and of sulfur in 14-carat white gold is shown.

Hard Metal Carbides and Cemented Tungsten Carbide

By Samuel L. Hoyt, Schenectady, N. Y.

(Trans., Inst. Met. Div., 9. 28,000 words)

Discussion of the hard metal carbides as a group of related metallic substances and of cemented tungsten carbide as the leading representative of the metallurgy of this group. Beginning with the periodic arrangement of the elements of hard metal carbides, the paper briefly reviews hardness measurements and X-ray investigation. The preparation of cemented tungsten carbide is described, its structure is examined and its properties are considered. Hot-pressed cemented tungsten carbide is compared with the cold-pressed product..

Cemented Tungsten Carbide

By L. L. Wyman and F. C. Kelley, Schenectady, N. Y.

(Tech. Pub. No. 354; Class E, Institute of Metals, No. 119)

This paper presents the preliminary results of an investigation of the action of the cobalt-rich binding constituent in cemented tungsten carbide alloys, showing that the original pure cobalt, which is added as a binder, becomes a solvent for the carbide grains which it holds together. This action results in a lower melting binder that is mobile, and which also provides a medium for solution and deposition of carbide grains. The authors investigated microscopically an entire series

of alloys from 3 to 100 per cent CO and show that the progressive changes throughout the series prove this solvent action, as well as show the reasons for many previously unexplained phenomena resulting from this cementing action.

Directed Stress in Copper Crystals

By C. H. MATHEWSON, New Haven, Conn., and KENT R. VAN HORN, Cleveland, Ohio
(Tech. Pub. No. 301; Class E, Institute of Metals, No. 109)

THE preparation of large crystals of copper is described, these crystals were then subjected to directed stress by squeezing in a vise. Every attempt to cause the slip on octahedral planes in a copper crystal to take the twinning direction (112) instead of the preferred direction (110) was unsuccessful, so that twinning by pure shear according to the diagram given by the authors appears to be out of the question. It is argued that twin crystals visible after annealing occur only after deformation of a complex nature in which slip on one set of octahedral planes is modified by simultaneous slip involving atoms in the same field of attraction on a crosswise set of octahedral planes; for example, (110) rows of atoms originally guided in their slip by adjacent rows move out of position by slip in another plane. This explanation is offered for the features shown in the figures which accompany the paper.

Thermal Conductivity of Copper Alloys

By CYRIL S. SMITH, Waterbury, Conn.
(Tech. Pub. No. 291; Class E, Institute of Metals, No. 102)

THIS paper describes the first of a series of experiments to determine the thermal conductivities of all commercial alloys rich in copper. It contains a complete review of previous work, and gives in detail new data on the copper-zinc alloys up to 50 per cent zinc. The thermal conductivity of the alloys decreases rapidly from 0.941 cal./sq. cm./cm./sec./° C. for pure copper to 0.285 for the saturated alpha solid solution, 39 per cent zinc. The appearance of the beta phase in the alloys causes an increase in conductivity and a very rapid decrease in the temperature coefficient. The decrease in thermal conductivity caused by adding zinc to copper is not as rapid as the decrease in electrical conductivity, although in general the two curves are similar in form.

The Thermal Conductivity of Copper Alloys

By CYRIL STANLEY SMITH, Waterbury, Conn.
(Tech. Pub. No. 360; Class E, Institute of Metals, No. 122)

THIS paper is a continuation of the work on the thermal conductivity of copper alloys described in the author's previous paper (A. I. M. E. TECH. PUB. 291, Feb., 1930). The thermal conductivity of copper (0.941 cal./sq. cm./cm./sec./°C.) is rapidly reduced by the addition of tin,

until with 10.41 per cent tin it is only 0.121 cal./sq. cm./cm./sec./°C. Phosphorus is ten times as powerful as tin, only 0.93 per cent phosphorus reducing the conductivity to 0.129 cal./sq. cm./cm./sec./°C. The electrical conductivity decreases more rapidly on alloying than does the thermal conductivity, and the Wiedemann-Franz-Lorenz ratio increases rapidly at first, but beyond 2.0 per cent tin or 0.15 per cent phosphorus remains almost constant. This break in the Wiedemann-Franz-Lorenz ratio curve has occurred in every system yet examined and is evidently of basic physical significance.

Certain Types of Defects in Copper Wire Caused by Improper Dies and Drawing Practice

By H. C. Jennison, Waterbury, Conn.

(Tech. Pub. No. 285; Class E, Institute of Metals, No. 97)

Two defects in wire, known respectively as "crowfeet" and "cuppy" wire, are known to result from dies of improper design or undesirable drawing practice; both cause brittleness of the wire. The author discusses die design and drawing practice and recommends procedure to overcome these defects.

Alpha Phase Boundary of Ternary System Copper-Silicon-Manganese

By Cyril S. Smith, Waterbury, Conn.

(Tech. Pub. No. 292; Class E, Institute of Metals, No. 103)

The equilibrium relations of the ternary alloys containing more than 90 per cent copper were determined by means of a series of cooling curves and the microscopic examination of a large number of annealed and quenched samples. The addition of manganese causes a depression of the temperatures of the reactions in the binary copper-silicon system, until at 2.5 per cent manganese the beta peritectic reaction has fallen to 760° C., at which temperature there is a quaternary reaction with Mn_2Si. The solubility of Mn_2Si decreases rapidly as the temperature falls, until at 450° C. it is less than 0.5 per cent with 4 per cent or more of either manganese or silicon.

Copper and Copper Alloys

By William H. Bassett, Waterbury, Conn.

(Min. & Met., December, 562. 3500 words)

A discussion of the uses of copper and its alloys under present industrial conditions. The physical properties are mentioned in this connection and in addition to brass, bronze and the cupronickels in general the following special alloys are considered: Ambrac, Everdur, Tempaloy and Avialite bronze. Some projected alloys with beryllium and silver are mentioned.

Sand-Cast Alloys of Copper

By J. W. Bolton and S. A. Weigand, Cincinnati, Ohio
(Min. & Met., January, 5. 750 words)

Developments during the year 1929.

Comparison of Copper Wire Bars Cast Vertically and Horizontally

By J. Walter Scott and L. H. DeWald, Chicago, Ill.
(Tech. Pub. No. 289; Class E, Institute of Metals, No. 100)

Vertically cast copper appears preferable to horizontally cast as wire bars produced in this way are free from wrinkles and oxygen segregation on long faces. It is also more dense, but regardless of initial density or method of casting, after cold working the density is approximately 8.91. Wire from vertical casts has a slightly lower tensile strength and slightly higher elongation after low-temperature annealing. The electrical conductivity also averages about 0.2 per cent higher. The authors believe that vertical casting will permit improvements in wire-drawing technique, which are discussed in detail.

A Theory Concerning Gases in Refined Copper

By A. E. Wells and R. C. Dalzell, Cambridge, Mass., and Roselle, N. J.
(Tech. Pub. No. 270; Class E, Institute of Metals, No. 93)

Part of the gases dissolved in molten copper evolve on solidification. The cuprous oxide in molten copper is colloidally dispersed and then adsorbs gases that are liberated on its agglomeration on solidification. The gases that cause porosity, CO_2, CO, nitrogen, water and hydrogen, are listed in the theoretical order of effect from changes in solubility at the melting point, but since nitrogen amounts to two-thirds the total volume it has the principal effect. Water and CO_2 are liberated by agglomeration of cuprous oxide and in practice this is the more important cause of porosity. SO_2 dissolves in molten copper, reacting to form sulfide and oxide. The quantities usually involved are too small to exert much effect upon porosity. As the sulfide decreases the dispersion of the oxide the oxygen content at tough pitch needs to be higher when sulfide is present.

The Alpha-beta Transformation in Brass

By Albert J. Phillips, Waterbury, Conn.
(Tech. Pub. No. 288; Class E, Institute of Metals, No. 99)

Conversion from beta to alpha in brass takes place with very great rapidity if there is no change in composition. Since the alpha structure produced upon quenching the 62 per cent copper alloy from the beta range was quite unexpected, special means of insuring its identity were resorted to: (1) A specimen was etched with ferric chloride, a reagent which darkens beta but leaves alpha light. This treatment did not change the appearance of the large alpha areas but merely inten-

sified the alpha-beta fringe at the crystal boundaries. (2) A polished and etched specimen was pinched in a vise producing slip lines characteristic of most metals crystallizing in the face-centered-cubic lattice. These slip lines showed the usual faultings through the twin bands present. Beta brass does not show slip lines upon slight cold working. (3) When a specimen was severely cold-worked it did not show the thin mechanical twin bands so easily produced in beta retained by quenching but deformed readily without cracking. (4) Upon annealing a specimen for 1 hr. at 550 C., there was no change in the appearance of the alpha. The alpha-beta fringe at the crystal boundaries, however, was converted entirely into alpha. This heat treatment brought about a decided drop in hardness which may have been due to the change at the crystal boundaries.

Oxides in Brass

By O. W. Ellis, Toronto, Canada

(Tech. Pub. No. 283; Class E, Institute of Metals, No. 96)

After discussing the general aspects of oxides in brass the author gives the results of a series of tests in which all the factors that could be controlled were kept the same with the exception of the nature of the charge. Tables 5 and 6 based on Tables 1, 2 and 3 show that a close connection exists between the oxide count and the nature of the charge. These also show that retention of the charge in the furnace between the first and second pours in the absence of flux increases the oxide count, but the presence of a flux tends to retain the oxide unchanged. Poling has a beneficial effect on a charge to which flux has been added. The author suggests that his method of "oxide count" may be adopted for routine factory control and merits further investigation.

Influence of Silicon in Foundry Red Brasses

By H. M. St. John, G. K. Eggleston, and T. Rynalski, Detroit, Mich.

(Tech. Pub. No. 300; Class E, Institute of Metals Division, No. 108)

The influence of silicon in a red brass alloy containing copper, tin, lead and zinc was investigated. It was found that very small percentages of silicon in such an alloy produced a coarsely dendritic structure, with large intercrystalline fissures. In the absence of lead this effect was not produced.

The effect of silicon could be counteracted by casting at low temperatures or in a chill. It could be entirely obscured by the use of substantial percentages of nickel. The silicon could be selectively oxidized by the use of sodium sulfate or barium sulfate as a flux.

Experimental evidence pointed to the conclusion that silicon tends to produce a large-grained structure by reducing the number of crystal nuclei formed or by prolonging the period of solidification. Carbon

monoxide plays no part in this result except that it permits the reduction of silicon in the melting furnace or protects it from oxidation if already present.

Internal Stress and Season Cracking in Brass Tubes

By D. K. Crampton, Waterbury, Conn.

(Tech. Pub. No. 297; Class E, Institute of Metals, No. 106)

After listing the points on which investigators are generally agreed the author lists three where agreement or adequate investigation is still lacking. He then describes his preliminary tests to find a simple method for comparing the internal stress in different tubes. The one adopted is described and then the results of a series of tests are given in tabulated form, showing the effect of variation in analysis, of type and degree of drawing reduction. The conclusions are: The general split-ring method for evaluating internal stress in tubes is only reliable when rather wide strips (namely, long sections of tubes) are used. The copper content of brass tubes has a profound effect on tendency to season crack. Tubes of 90 per cent copper or over are practically immune and those over 80 per cent copper are fairly so. Tubes in the high-brass range are strongly susceptible to season cracking. Tubes of 60 per cent copper are decidedly the most susceptible. Iron and lead have no practical effect on season cracking. Lead, however, has a very powerful effect on fire cracking. Tin has a slight but distinctly protective effect on season cracking. The reciprocal of the time in minutes to crack in standard $HgNo_3$ solution is a fairly reliable criterion of the tendency to season crack. The tendency to season crack increases directly with intensity of internal stress. Total immunity is obtained when the circumferential stress by the method used is below approximately 12,000 lb. per sq. in. In high-brass tubes the intensity of internal stress and the tendency to season crack are increased by increase of wall thickness in proportion to diameter, by hollow sinking instead of drawing over a plug, by increase of diameter reduction; are independent of Rockwell hardness or other physical properties; are decreased by increasing area reductions. With properly designed operations it is possible to draw practically any size of brass tube to any degree of hardness and have it free of tendency to season crack. The difficulty of accomplishing this, however, increases considerably with very thick-walled tubes. Tubes so drawn obviously do not require relief annealing to be safe from season cracking.

Stress-Corrosion Cracking of Annealed Brasses

By Alan Morris, Bridgeport, Conn.

(Tech. Pub. No. 263; Class E, Institute of Metals, No. 91)

After reviewing the literature on this subject the author describes the apparatus used to test samples prepared from ½-in. round rods by exposing them to attack by ammonia. The results, which are

given in detail by tabular summaries and charts, showed that coarse grain in so-called "high brasses" appears to lower the resistance of the piece to stress-corrosion attack. Lead and tin in an alpha brass tend also to make the material a little less resistant to this form of attack. The resistance of a Muntz metal and naval brass (and probably of manganese bronze) is materially increased by quenching from a reasonably low annealing temperature. The author does not feel that these tests constitute more than a general reconnaissance of the field, and they are presented in the hope that their discussion will present suggestions that will serve as a guide to further work of this nature.

Effects of Oxidation and Certain Impurities in Bronze

By J. W. Bolton and S. A. Weigand, Cincinnati, Ohio

(Tech. Pub. No. 281; Class E, Institute of Metals, No. 94)

Porosity due to incipient shrinkage is influenced by the furnace atmospheres. In this paper the bad influences of actual oxidation are apparent in incipient shrinkage, lowered strength, sluggishness of metal, and zinc loss. These remarks apply to metals containing only traces of impurities. When melted in a crucible under practically neutral furnace atmospheres, the impurities silicon, sulfur and aluminum have effects which do not resemble the usual atmospheric effects. In some cases these impurities have a deleterious influence. Even small percentages of aluminum change the (skin) color of the alloy, and modify its crystallization characteristics. Larger percentages make it weak and brittle, with extremely coarse grain. The inclusions are incidental, if interesting. The hardness is reduced to 57, from 60 to 65. The effect is not accompanied by as low specific gravity as is encountered in gassed metal. Silicon also appears to go into solid solution and modifies the crystallization characteristics of the metal. When melted under neutral atmospheres, no inclusions attributable to presence of silicon are discernible. While silicon in minute amounts is not dangerous, over 0.05 per cent should be avoided in commercial practice. In alloys high in lead this element may need to be held even lower. The action of sulfur is less marked, but in the writers' opinions percentages over 0.05 per cent are not desirable.

Effect of Certain Alloying Elements on Structure and Hardness of Aluminum Bronze

By Selma F. Hermann, Dayton, Ohio, and Frank T. Sisco, New York, N. Y.

(Tech. Pub. No. 365; Class E, Institute of Metals, No. 123)

The alloys studied in this paper include five series of aluminum bronzes with additional alloying elements, namely, nickel, iron, manganese, cobalt and silicon. Analogous compositions and analogous heat treatments were used so that there would be some basis for comparison among the specimens. Alloys with 8, 10 and 12 per cent alumi-

num were selected, so that the effect of the alloying elements on the alpha, alpha plus beta, beta and eutectoid phases could be noted. Hardness tests and grain size measurements were made wherever possible. Nickel and iron increase the general hardness somewhat; cobalt, manganese and silicon increase it very materially. The coefficient of equivalence of nickel with respect to the aluminum content is small; in the case of iron, it is zero; for manganese and silicon it is quite appreciable; and for cobalt it is negative. All the alloying elements introduced one or more special constituents, which in general were almost unaffected by heat treatment.

Melting Bearing Bronze in Open-flame Furnaces

By Ernest R. Darby, Detroit, Mich.

(Tech. Pub. No. 302; Class E, Institute of Metals, No. 110)

Attention is called to the normal chemical actions which take place in open-flame furnaces used in the melting of bronze. Oxidizing, neutral and reducing atmospheres are considered with reference to their effects upon the chemical compositions and physical qualities of the metal melted. Control of furnace atmosphere, together with close observation of physical properties as determined by routine laboratory tests, makes possible the production of high-grade castings from classes of raw material frequently considered inferior. The extent to which refining may be carried economically is dependent on the quality of the raw material, the kind of furnaces used, the nature of the castings to be made and the general foundry layout.

Studies Upon the Widmanstätten Structure. I. Introduction. The Aluminum-silver System and the Copper-silicon System

By Robert F. Mehl and Charles S. Barrett, Washington, D. C.

(Tech. Pub. No. 353; Class E, Institute of Metals, No. 118)

When a new phase appears in an alloy upon cooling as a result of a decreasing solid solubility it takes a characteristic form, designated in steels and meteorites as the Widmanstätten figure. This figure demonstrates a relationship in crystallographic orientation between the lattice of the newly formed (or precipitated) phase and the lattice of the parent solid solution. Such a relationship must illustrate the operation of some basic crystallographic mechanism in solid-solid transitions of this type. The present paper is the first of a series on the general subject of the Widmanstätten structure in which various structures will be subjected to crystallographic analysis with the object of studying this crystallographic mechanism. Previously published research on the general subject is critically reviewed. It is thought desirable to restrict the term Widmanstätten figure to those alloy structures in which the separate phases can be identified unmistakably. Structures suggesting the Widmanstätten figure but not fulfilling this requirement, like that of martensite, are termed quasi-Widmanstätten

figures. The Widmanstätten figure resulting from the precipitation of the γ phase from the Al-rich terminal δ solid solution in the Al-Ag system is analyzed, and it is shown that the γ phase forms plates parallel to the octahedral or (111) planes in the face-centered cubic δ phase and that the basal plane in the hexagonal γ phase is parallel to this plane with its atoms so arranged as nearly to coincide in position with the atoms on the (111) plane in the δ phase. Analysis of the figure resulting from the precipitation of the γ phase from the face-centered cubic α solid solution in the Cu-Si system shows that the γ phase precipitates as plates parallel to some twelve-family phase. This analysis demonstrates the limitation of Osmond and Cartaud's generalization that a precipitate should deposit parallel to the plane of densest atomic packing. A tentative theory of the mechanism leading to the formation of the Widmanstätten figure is outlined.

Aluminum and Aluminum Alloys

By Sam Tour, New York, N .Y.

(Min. & Met., January, 7. 1450 words)

Summary of developments during 1929 in production of large structural shapes, aluminum paint for prevention of corrosion, all-metal aircraft, X-ray inspection of aluminum castings and forgings, welding and research work. A short bibliography is included.

Equilibrium Relations in Aluminum-magnesium-silicide Alloys of High Purity

By E. H. Dix, Jr., F. Keller and R. W. Graham, New Kensington, Pa.

(Tech. Pub. No. 357; Class E, Institute of Metals, No. 121)

The solid solubility relations in a series of aluminum-magnesium-silicide alloys made from extremely high purity aluminum (99.96 per cent) and containing from 0.14 to 2.37 per cent of the compound Mg_2Si were investigated. Since the alloys were readily workable the solubility determinations were made on sheet specimens, with the result that equilibrium conditions were obtained with much shorter heating periods. A solid solubility of 1.85 per cent Mg_2Si was obtained at the eutectic temperature (595° C.) and the solubility decreased with decreasing temperature until at 200° C. it was less than 0.27 per cent. Mechanical properties were obtained on the wrought alloys in the hard rolled, annealed, quenched, and quenched and artificially aged tempers. A previous investigation of the solubility relations of Mg_2Si in aluminum was made by Hanson and Gaylor and their results indicated a solid solubility of 1.6 per cent Mg_2Si at the eutectic temperature and less than 0.6 per cent at 30° C. The results of the present investigation do not check these values, and the difference may result from the fact that the former investigators used aluminum of much lower purity and did not use long enough heating periods, especially at the lower temperatures, to obtain equilibrium conditions.

Aluminum-silicon-magnesium Casting Alloys

By R. S. Archer and L. W. Kempf, Cleveland, Ohio

(Tech. Pub. No. 352; Class E, Institute of Metals, No. 117)

In 1921 commercial production was begun on aluminum-alloy-castings having a higher combination of strength and ductility than had previously been available, the improved properties being obtained largely by the use of a new process of heat treatment.[1] The alloy used for this purpose contains approximately 4 per cent copper, with suitable control of iron and silicon. Castings are strengthened and made more ductile by a high-temperature solution heat treatment, and additional hardening may be obtained if desired by a low-temperature precipitation heat treatment. The minimum properties of sand-cast test specimens, for three conditions of heat treatment, are as follows:

Alloy Designation	Precipitation Treatment	Tensile Strength, Lb. per Sq. In.	Elongation, Per Cent
195–4	None	28,000	6
195–10	Short	30,000	3
195–16	Long	36,000	0

Higher properties are obtained on chill castings of similar composition and heat treatment. The extensive and successful use of these castings in the past nine years has demonstrated the need for a product having mechanical properties of the kind described.

It is shown in the present paper that mechanical properties comparable with those of the heat-treated 4 per cent copper alloy castings can be obtained with heat-treated aluminum castings containing 3 to 13 per cent silicon, 0.1 to 0.5 per cent magnesium, and 0 to 1.0 per cent copper. Machinability is much better than that of the binary aluminum-silicon alloys, although probably somewhat inferior to that of the 4 per cent copper alloy. The advantages of the aluminum-silicon-magnesium alloys include excellent casting qualities, high resistance to corrosion, high thermal and electrical conductivity, low specific gravity, and low thermal expansivity. These alloys are not to be considered entirely as substitutes for the 4 per cent copper alloy, however, but rather as new materials having characteristic properties which, it is believed, will extend the fields of application of high-strength aluminum alloy castings.

Equilibrium Relations in Aluminum-antimony Alloys of High Purity

By E. H. Dix, Jr., F. Keller and L. A. Willey, New Kensington, Pa.

(Tech. Pub. No. 356; Class E, Institute of Metals, No. 120)

The solid solubility relations of antimony in aluminum were determined on a series of alloys, containing from 0.10 to 1.14 per cent antimony, made from extremely high purity aluminum (99.96 per cent). The eutectic concentration and eutectic temperature were also deter-

[1] R. S. Archer and Zay Jeffries: Aluminum Castings of High Strength, *Proc.* Inst. Metals Div., A. I. M. E. (1927) 45.

mined. The solubility relations were obtained by microscopic examination of chill cast samples of the alloys after they had been heated for long periods to obtain equilibrium conditions. The results of the investigation indicate a solid solubility of less than 0.10 per cent antimony at temperatures as high as 645° C. The eutectic concentration lies at about 1.1 per cent antimony and the eutectic temperature is 657° C.

Constituents of Aluminum-iron-silicon Alloys

By William L. Fink and Kent R. Van Horn

(Tech. Pub. No. 351; Class E, Institute of Metals, No. 116)

Diffraction patterns were obtained from aluminum-iron-silicon alloys which had been subjected to prolonged annealing treatments at 550° and 560° C. In some instances patterns were made from constituents which had been concentrated by electrolytic separation. The X-ray results confirm the previous work of Dix and Heath. Two ternary constituents, α(Fe-Si) and β(Fe-Si) were found in the range of compositions investigated. The constituent α(Fe-Si) appeared to be a solid solution of silicon in the binary compound $FeAl_3$. The diffraction data indicated that β(Fe-Si) is a ternary compound which may have an excess of aluminum in solid solution, causing an expansion of the parent lattice.

Modulus of Elasticity of Aluminum Alloys

By R. L. Templin and D. A. Paul, New Kensington, Pa.

(Tech. Pub. No. 366; Class E, Institute of Metals, No. 124)

The authors of this paper present the results of a preliminary investigation describing the variation of the tensile modulus of elasticity (E) with the addition of alloying metals to aluminum. A brief description is given of the apparatus used and of the method of determining E from stress-strain data.

The results of the investigation are summarized in the following conclusions:

1. A general increase of modulus of elasticity results with the addition of iron, silicon, copper, nickel, or manganese to aluminum.
2. In certain of the commercial aluminum alloys the modulus of elasticity is increased as much as 20 to 30 per cent by the presence of considerable amounts of the above metals.
3. The results obtained indicate that copper, iron and nickel produce changes somewhat in the order of their respective modulus values; nickel having the greatest effect and copper the least.
4. The addition of silicon to aluminum results in an increase of modulus of elasticity with a decrease in the density of the metal.
5. In heat-treated aluminum-magnesium alloys the modulus of elasticity does not decrease until magnesium is present in amounts greater than about 12 per cent; in unheat-treated alloys the decrease occurs in the region between 6 and 10 per cent magnesium.

6. It may be concluded from 5 that magnesium held in solid solution in aluminum does not have as great an influence on the modulus of elasticity as when it is present as a free constituent.

Quenching Alclad Sheet in Oil

By HORACE C. KNERR, Philadelphia, Pa.
(Preprint; Class E, Inst. of Metals Div. 2400 words)

THE rate of cooling of duralumin sheet occasioned by quenching in various mediums such as water, oil and air, does not greatly influence its tensile properties but does influence its resistance to corrosion, the more rapid cooling producing the best resistance. Distortion during quenching is greatest in the rapid-cooling mediums.

Alclad sheet owes its tensile properties to the internal body of the duralumin and its corrosion resistance to the surface coating of pure aluminum. The latter is not affected by the quenching treatment. Therefore it should be possible to quench Alclad sheet in oil or other mild quenching medium, avoiding distortion, obtaining satisfactory tensile properties and retaining satisfactory corrosion resistance. This has been confirmed experimentally.

Rolling of Aluminum Structural Shapes at the Massena Plant of the United States Aluminum Co.

By W. F. BOERICKE, New York, N. Y.
(Min. & Met., April, 222. 2000 words)

BRIEF description of the largest aluminum rolling mill in the world, which is also the first mill in the United States to roll aluminum exclusively.

Lead and Lead Alloys

By G. O. HIERS, Brooklyn, N. Y.
(Min. & Met., January, 5. 600 words)

SUMMARY of improvements developed during 1929, with a short bibliography.

Distribution of Lead Impurity in a Copper-Refining Furnace Bath

By J. WALTER SCOTT and L. H. DEWALD, Chicago, Ill.
(Tech. Pub. No. 290; Class E, Institute of Metals, No. 101)

LEAD occurs more or less uniformly distributed through the bath, but as oxidation proceeds there is a slight tendency for it to concentrate toward the surface. Initial reduction of lead content is easier when large amounts are present; the oxygen content increases progressively during the oxidation period and decreases with depth. To remove lead efficiently by fire refining methods the lead must be in the oxidized condition so it may unite with the carrier slag, and a high degree of agitation must be maintained so that every particle of copper

containing lead will have an opportunity to come into contact with the slag.

Nickel in 1929

(Min. & Met., January, 8. 800 words)

REVIEW of improvements in the purity of nickel alloys and the use of nickel for corrosion resistance.

Monel Metal and Nickel Foundry Practice

By E. S. WHEELER, Bayonne, N. J.

(Tech. Pub. No. 298; Class E, Institute of Metals, No. 107)

MONEL metal and nickel both being harmfully susceptible to the actions of gases all possible care should be taken to permit their easy escape. For the same reason the use of gas producing ingredients in molding materials should be reduced to a minimum. Manganese and sulfur are both detrimental, manganese causing the metal to cut the sand badly and, in addition, causes heavy shrinks, while sulfur in excess of 0.030 per cent makes the metal hot—short and brittle. A good grade of refractory molding sand should be used for the molds. Because of their high shrinkage Monel metal and nickel castings are likely to develop shrinks and pulls which can often be overcome by a change of molding practice, the use of heavier fillets, a change of mixture, the elimination of all or part of the scrap used in the charge, or a change in the core when present to permit the casting to solidify without any strain being set up by the excessive hardness of the core. Scrap should be kept down to a minimum and should not exceed 25 per cent of heavy scrap or 10 per cent of light scrap. The use of excessive quantities of scrap is frequently responsible for heavy pulls and sometimes causes porosity due to the oxidation of the silicon, which is usually present, and to the evolution of gas picked up during remelting. Melting should be accomplished as quickly as possible, but not at so fast a rate as to oxidize the metal. In the crucible pits about 2 hours should be necessary to melt Monel metal in crucibles containing 175 lb. of metal, and slightly longer for nickel. This melting rate applies to a hot furnace; obviously, the first melt will require a slightly longer time. The most important factor necessary for the production of good Monel metal or nickel castings is suitable equipment. Without the proper furnaces no amount of good molding or good foundry practice will produce satisfactory castings.

Expansion Properties of Low-expansion Fe-Ni-Co Alloys

By HOWARD SCOTT, East Pittsburgh, Pa.

(Tech. Pub. No. 318; Class E, Institute of Metals, No. 112)

IN response to technical demands for metals having low expansion at high temperatures, the possibility of improving the low-expansion nickel steels by alloying additions was investigated. Brace found that cobalt

may be added with distinct improvement, but not without complications. Study of a preliminary series of Fe-Ni-Co alloys revealed the nature of the complications and permitted an explicit statement of the problem of finding optimum compositions. Experimental attack on this problem yielded data which were condensed to form equations giving the expansion properties in terms of composition for a useful range of compositions. From these equations the optimum compositions of commercially feasible alloys were deduced. The properties calculated for the optimum composition have received excellent verification from tests of alloys prepared to these compositions.

Influence of Cyclic Stress on Corrosion

By D. J. McAdam, Jr., Annapolis, Md.

(Tech. Pub. No. 329; Class E, Institute of Metals, No. 113)

This paper discusses the influence of cyclic stress on corrosion of carbon and ordinary alloy steels, corrosion-resisting steels, monel metal and aluminum alloys. The damage due to corrosion is estimated by comparing the fatigue limit of the previously corroded specimen with the endurance limit of the metal. Specimens are corroded with or without cyclic stresses of various frequency, and for various times. The corroded specimen is then oiled, its fatigue limit is determined and compared with the endurance limit of the uncorroded material. The lowering of the fatigue limit represents the "damage" due to corrosion.

The results of the investigation are expressed in diagrams of various types illustrating the relationship between corrosion-stress, time, number of cycles and either total or net damage. By "net damage" is meant the total damage less the damage that would be caused in the same time by stressless corrosion. The effect of cyclic stress on corrosion is measured by net damage rather than by total damage. Relationship of net damage to corrosion-stress, time and number of cycles, may be represented on a logarithmic scale by nearly straight lines.

Even very low corrosion stresses have noticeable effect in accelerating the damage due to corrosion. For steels and aluminum alloys, alternating stress of only 3000 lb. per square inch causes considerable net damage. Stress as low as 2000 lb. has appreciable effect. The effect of even lower stress could probably be detected, if experiments could be continued long enough.

For corrosion-resisting alloys, stresses far below the corrosion-fatigue limits, as obtained by previously described short-time tests, cause decided net damage. Tentative conclusion is drawn that any stress cycle, however small the stress range, accelerates corrosion pitting.

The total damage to steels, when corroded with or without cyclic stress, proceeds at a gradually decreasing rate. The total damage to monel metal or corrosion-resistant steels, when corroded with or with-

out cyclic stress, proceeds at a continually accelerated rate. The damage to aluminum alloys under stressless corrosion proceeds at a gradually decreasing rate. The damage to aluminum alloys, when corroded under cyclic stress, may either proceed at a continually increasing rate, or may proceed first at a gradually decreasing rate and then at a continually accelerated rate. Reasons for these differences in behavior are discussed. Discussion, September, 427.

Corrosion of Alloys Subjected to the Action of Locomotive Smoke

By F. L. Wolf, Mansfield, Ohio

(Tech. Pub. No. 293; Class E, Institute of Metals, No. 104)

This paper gives a long list of specimens on which tests were started in 1923 and are still in progress, with the cooperation of the N. Y., N. H. & H. R.R., the Boston and Maine, and the Pennsylvania. The test specimens were suspended in the smoke-jacks of roundhouses, one specimen being removed and examined at six-month intervals. While the results tabulated are not entirely conclusive they give an indication of the various types of alloys commercially available for overhead work in railroad construction. They also indicate that some alloys which stand up well under ordinary corrosion are not able to withstand the corrosive action of locomotive smoke.

Working Properties of Tantalum

By M. M. Austin, Chicago, Ill.

(Tech. Pub. No. 278; Class E, Institute of Metals, No. 105)

Metallic tantalum was used as filament in incandescent lamps in 1906, but only within the last five years has it been available in sufficient quantity for general application. It has a high-melting point and strong resistance to corrosion. It can be subjected in a remarkable degree to cold working. A bar of pure tantalum 0.4 in. thick can be rolled to a sheet 0.001 in. thick without heating or annealing. Some annealing at 500 C. is necessary to draw wire to 0.001 in. diameter. The paper describes other properties of the metal, particularly striking demonstrations of block movement during plastic flow.

X-ray Notes on the Iron-molybdenum and Iron-tungsten Systems

By E. P. Chartkoff and W. P. Sykes, Cleveland, Ohio

(Tech. Pub. No. 307; Class E, Institute of Metals, No. 111)

This paper reports measurements of changes in lattice parameters in solid solutions. Precipitation from the solid solution and the accompanying changes in hardness were studied, as well as the formation of the intermetallic compound phases upon sintering a mixture of metal powders.

The Application of X-rays to Development Problems Connected with the Manufacture of Telephone Apparatus

By M. Baeyertz, Chicago, Ill.

(Tech. Pub. No. 349; Class E, Institute of Metals, No. 115)

This application of X-rays falls into two categories: (1) radiography and (2) diffraction patterns. Radiography as used in the development of equipment and in studies of materials is discussed by means of typical examples. One use of diffraction patterns is mentioned.

Influence of Casting Practice on Physical Properties of Die Castings

By Charles Pack, New York, N. Y.

(Tech. Pub. No. 346; Class E, Institute of Metals, No. 114)

It has been the prevailing belief that the physical properties of die castings are entirely dependent upon the composition of the alloy used. Deviations in the physical properties of die castings have been promptly attributed to alloy variations and it has been rather difficult for the plant metallurgist to enlist the cooperation of those responsible for machine and die design, in his efforts to produce die castings of better physical properties. This paper helps to emphasize the fact that the production of good die castings that will meet maximum physical requirements is not entirely within the control of the plant metallurgist, but that this responsibility must be shared equally by the mechanical and metallurgical divisions of the plant. The present die-casting industry has been built upon the principle that the mechanical phase was predominant. Mechanics designed the die-casting machine and the die for use with the machine, leaving it to the metallurgist to provide suitable alloys. Under these conditions, the metallurgist was greatly limited in his choice of alloys. There is a growing tendency to recognize the importance of the metallurgical phase of the industry in future in designing die-casting machines. Future development of die-casting equipment may be expected to comply with metallurgical practice.

An Air-hardening Copper-cobalt Alloy

By Cyril Stanley Smith, Waterbury, Conn.

(Min. & Met., April, 213. 3000 words)

Experiments on an alloy containing 3.6 per cent of cobalt show that rapid quenching renders the alloy extremely soft, but slower cooling results in an increasingly hard alloy until very slow rates of cooling are reached, when progressive softening occurs. Curves are reproduced showing the hardness after cooling at various speeds from 900° C. and also the hardness produced by reannealing, at a series of temperatures, specimens which had been both rapidly quenched and air cooled.

It was found that alloys which had been annealed in air had a layer of metallic "subscale" underneath the usual black and red scale. This

"subscale" consists of fine particles of cobalt oxide in a matrix of pure metallic copper with the original twinned grain structure, and indicates an appreciable solubility of oxygen in copper. Discussion, June, 322.

Destructive and Non-destructive Tests of Welds

By A. B. KINZEL and J. R. DAWSON, Long Island City, N. Y.

(Min. & Met., June, 308. 6300 words)

DESCRIPTION of methods for testing a weld immediately after it is made without removing samples to a laboratory, and of new non-destructive tests.

Arc Welding in Industry

By H. M. FRENCH, Schenectady, N. Y.

(Min. & Met., July, 352. 2900 words)

SHORT description of the kinds of arc welding, with a discussion of their uses and advantages.

Non-ferrous Secondary Metals

By E. R. DARBY, Detroit, Mich.

(Min. & Met., January, 9. 1350 words)

REVIEW of the more important developments in the handling and use of these metals, with a short bibliography.

The Present Radium Situation

By R. B. MOORE, Lafayette, Ind.

(Min. & Met., February, 91. 2200 words)

THIS paper deals with the deposits of radium-bearing ore, the uses of radium, the cost of its production and the amount of production. Discussion in April, 225, and August, 380.

Technical and Commercial Trends in the Junior Metal and Mineral Industries

By G. C. RIDDELL, New York, N. Y.

(Min. & Met., January, 43. 8700 words)

A PEAK in the high standard of living in the United States, accompanied by notable expansion in aviation, radio and automatic mechanical equipment generally, made 1929 a year of intense activity in the minor minerals and metals. In addition to information on all the rarer minerals, the article summarizes activities in electronic engraving, all-metal aircraft and sulfuric acid.

Technology of the Precious Metals

By GEORGE F. KUNZ, New York, N. Y., and EDMUND M. WISE, Bayonne, N. J.

(Min. & Met., January, 10. 3200 words)

A RÉSUMÉ of the uses of precious metals and their alloys, with a selected classified bibliography.

Progress in Theoretical Metallurgy during 1929

By R. S. Dean, Washington, D. C.
(Min. & Met., January, 4. 1100 words)

Review of work on heat treatment and in regard to the chemical explanation of anomalous mechanical properties in metals, accompanied by a short bibliography.

COAL AND COKE

Coal in 1929

By Howard N. Eavenson, Pittsburgh, Pa.
(Min. & Met., January, 17. 3100 words)

Research and service work to promote satisfaction of the consumer and efficiency in utilization of coal were notable features. Increase in use of synthetic products and natural gas. Classification progressing. Low-temperature carbonization a commercial process.

What's Ahead in Coal Production

By J. B. Dilworth, Philadelphia, Pa.
(Min. & Met., November, 514. 550 words)

Conclusions of a paper presented at the National Coal Association Meeting, Detroit, Mich., Oct. 16, 1930.

Stabilization of Credit and Operation in the Coal Industry

By Frank Haas, Philadelphia, Pa.
(Min. & Met., February, 81. 4300 words)

A revolutionary proposal to bring about stabilization of production and prices through public ownership of the fee to coal deposits. Discussion of this paper appeared as follows: April, 226; October, 497.

How to Help the Coal Industry

By C. E. Bockus, Dante, Va.
(Min. & Met., March, 182. 1600 words)

Engineering can help. There are probably still ways in which methods of mining and preparation can be bettered. The still greater problem of the industry is to find a sale for its product at a price that will return cost plus a reasonable margin of profit on the large investment, and that amount must provide proper allowance for the surplus capacity that is absolutely necessary to even up the saw-teeth in the curve of demand.

The Mine Official as a Teacher

By E. A. Holbrook, Pittsburgh, Pa.
(Min. & Met., June, 306. 2000 words)

Suggestions to foremen in coal mines, which are applicable to all foremen, regarding the correct way in which to instruct the men under their direction.

Ventilation Problems at the World's Largest Coal Mine

By Henry F. Hebley, Chicago, Ill.
(Trans., Coal Div., 9. 7000 words)

New Orient mine produces 12,000 tons per 8-hr. day and has an allotted acreage of 15 square miles. Study of air requirements for the life of this mine proved that it was advisable to change from a four to a six-entry system, and to place an auxiliary air shaft in the center of the property. It was found that the most economical arrangement would be to sink separate circular shafts (15 ft. inside diameter and concrete lined) for the upcast and downcast air, one of which can be deferred for several years.

Subsidence in Thick Freeport Coal

By John M. Rayburn, Pittsburgh, Pa.
(Trans., Coal Div., 144. 1600 words)

A precise record by surveys of the subsidence, 1924-1928, in systematic retreating work in the extraction of pillars by a method equivalent to retreating longwall, at the mine of the Allegheny-Pittsburgh Coal Co., Springdale, Pa.

Flexible Roof Supports in Coal Mines

By E. C. Weichel, Scranton, Pa.
(Min. & Met., June, 292. 3200 words)

This permanent lining is formed of concrete blocks specially designed to meet the specific conditions required in each case. The patents are held by Hans Schaefer of Essen, Germany. The American licensee is Richard Howells of Scranton, Pa.

Shaker-Chute Mining—Northern Anthracite Field

By Kenneth A. Lambert, Scranton, Pa.
(Tech. Pub. No. 359; Class F, Coal and Coke, No. 38)

An adaptation of shaker-chute for recovery of pillars in caved areas has been successfully accomplished in the northern anthracite field. The use of shaker-chutes for transporting coal from working faces to loading point allows considerable reduction in size of opening which has to be made and maintained, resulting in increased speed of driving openings, decreased volume of waste material which has to be handled and decrease in cost of timbers necessary for maintenance of openings.

The working faces are advanced within the pillars to the outer limit of working area and remaining pillar coal recovered on the retreat. Small-sized shaker-chutes with individual drives deliver coal to main shaker-chute on the counter gangway extended across pillars. Ventilation of working places is accomplished by means of a small blower fan delivering through vent tubing provided with "Y" connections to split ventilation between working places.

Cost of mining by this method shows $2.27 per ton, compared with $7.80 per ton for ordinary hand mining methods.

Barrier Pillar Legislation in Pennsylvania

By George H. Ashley, Harrisburg, Pa.

(Tech. Pub. No. 277; Class F, Coal and Coke, No. 30)

Pennsylvania law formerly provided that protective barrier pillars between neighboring coal mines should have a minimum thickness based on the "water head" created in the event that one of the mines was abandoned. In compliance it was necessary to leave several hundred feet of pillar in many instances. Believing that this involved an unnecessary waste of coal the industry prevailed upon the governor to appoint a commission to revise the law in view of sound engineering principles. Hearings were held and a report was made in which was presented a formula for determining the minimum thickness based on the thickness of the coal bed and of the cover. The resulting act was signed by the governor in April, 1929.

Safety in Mining

By John T. Ryan, Pittsburgh, Pa.

(Min. & Met., October, 489. 1800 words)

Improved results shown by a group of coal-mining companies with well-organized safety departments furnish the answer to the question as to whether safety pays.

Loss in Agglutinating Power of Coal Due to Exposure

By S. M. Marshall, H. F. Yancey and A. C. Richardson, New York, N. Y., Seattle, Wash., and Tuscaloosa, Ala.

(Tech. Pub. No. 317; Class F, Coal and Coke, No. 32)

Results are given of some experiments to determine the loss of agglutinating power of three weekly coking coals with time, using the method which was described in February, 1929, by Marshall and Bird. The data indicate that a high oxygen coal when exposed freely to the atmosphere will absorb more oxygen and will decrease in agglutinating power. Insufficient data were available on the change in oxygen content to permit a continuous comparison to be made between the rate of oxygen increase and the rate of agglutinating power decrease. But the results apparently show that the increase in oxygen is accompanied by a definite and reasonably proportionate decrease in agglutinating strength.

Control of the Quality of Shipped Coal

By R. G. Baughman, Linton, Ind.

(Preprint; Class F, Coal and Coke. 2000 words)

Requirements of the coal market, methods of stripping an impure coal bed, methods of washing and testing, and a summary of results obtained in the washery during the first three months of 1930.

Dry Cleaning of Coal in England

By KENELM C. APPLEYARD, Birtley, England
(Tech. Pub. No. 374; Class F, Coal and Coke, No. 42)

THIS paper was referred to at some length in MINING AND METALLURGY for October, 1930, p. 485. It makes the point that dry cleaning is a process, not a machine, and discusses problems that are collateral to the main process. Aspiration is important. Dust collecting and disposal, screens and screening, vibration of buildings, bunkering, blending of screens, size segregation and breakage, alternative methods of machine arrangement and costs and technical results are the most important of the topics discussed.

Heat Drying of Washed Coal

By S. M. PARMLEY, Pittsburgh, Pa.
(Tech. Pub. No. 376; Class F, Coal and Coke, No. 43)

THIS paper deals with the operating factors relating to the heat-drying of 0 by 3/8-in. washed coal, as practiced by the Pittsburgh Coal Co. at its Champion No. 1 preparation plant, rather than with a theoretical discussion of heat-drying principles. The moisture content of the wet product is first reduced by centrifugal dryers and vacuum filters to 7.5 to 9.5 per cent and then heat-dried to 3.0 per cent moisture by means of indirect, coal-fired, rotary type dryers. The moisture on the −3/8-in. product, as shipped from a wet-cleaning plant, is lower than the moisture of the −3/8-in. raw coal. The difficulty encountered in the heat-drying of fine coal with a large percentage of −48-mesh material is stressed, and data given. Data are given relative to mechanical properties of dryers, discharge-moisture control, heat and air regulation, and coal and power consumption. The operating methods and rules are given for the elimination of fire hazards. There are seven heat dryers of various capacities operating at three plants, with a total potential drying capacity of 210 tons per hour.

Determination of Shapes of Particles and Their Influence on Treatment of Coal on Tables

By H. F. YANCEY, Seattle, Wash.
(Tech. Pub. No. 341; Class F, Coal and Coke, No. 37)

THIS PAPER describes a method of screening in which particles are separated into three shapes—cubical, prismatic and flat—by means of two sets of sieves, one with square and the other with rectangular or slotted openings. The proportion of cubical particles decreases and the proportion of flat particles increases with increase in specific gravity of the components of raw coal examined. Numerical expression is given to the more flaky character of bone and refuse—a commonly observed fact. The screening method was also used to study the influence of shape of particles in the treatment of a coal on a coal-washing table. The data show that cubical shapes in the coarse sizes are discharged first

and that flaky shapes of the same specific gravity and square-mesh screen size are carried farther out on the table deck before they are discharged. In the fine sizes this condition is reversed; flaky material is discharged in greater proportion than cubical material. The predominating tendency, cubes ahead of flakes or flakes ahead of cubes, may depend upon the proportion of coarse and fine sizes in the table feed. The coal used in the work described consisted mainly of coarse sizes and showed cubes to be discharged first. The tendencies revealed by this investigation have been discussed in some detail in an effort to analyze the influence of shape. The discussion may magnify the importance of shape. In general it must be concluded that shape of particle is a factor of minor importance in tabling unsized coal, in so far as the overall efficiency of the process is concerned. Size and, of course, specific-gravity difference are the major factors.

Coal Preparation Problems in the Illinois Field

By David R. Mitchell, Urbana, Ill.

(Tech. Pub. No. 362; Class F, Coal and Coke, No. 40)

This paper directs attention to some of the outstanding preparation problems in this field. A fundamental study of the physical and chemical characteristics of these coals is urged as the best means for solving these problems. Tables of tests made are presented to illustrate problems of screening, concentration of impurities and fusain, hand picking, ash and sulfur removal and float and sink testing.

Control of Chance Cone Operation

By John F. McLaughlin, Scranton, Pa.

(Tech. Pub. No. 361; Class F, Coal, No. 39)

Operating with water and silica sand of 2.63 specific gravity produces a fluid mass of approximately 1.72 specific gravity. This, or a lower gravity, is maintained by the volume of agitation water used, which is controlled by the operator's judgment after placing gravity balls in the fluid mass. The operator is not positive of the volume of water admitted as several of the sprays may be blocked and the specific gravity of the fluid mass, as determined by the gravity ball, may be inaccurate due to the ball being held in suspension by a floating mass. These inaccuracies result in uneven preparation, condemnation, or excessive losses to the bank.

A more positive means of determining the specific gravity of the fluid mass is to sample the final slate of the separating cone at 12-minute intervals for the hour period, and place it in a zinc chloride solution of the same specific gravity as it is desirous of maintaining in the separating cone. The amount of float material in the final state will determine if the specific gravity of the fluid mass in the separating cone is higher or lower than that desired.

To operate this system of control properly the feed should be prop-

erly regulated to prevent overload; sand losses held to a minimum, and silt removed throughout the operating day.

Sink and float tests and amount of water used for agitation should be tabulated hourly at cone operator's station so as to guide operating officials in efficient plant operation.

The Mechanical Preparation of Pocahontas Coals—Some Factors in the Problem

By J. R. CAMPBELL, Scottdale, Pa.

(Tech. Pub. No. 363; Class F, Coal, No. 41)

IN the paper on the above subject the author has covered the following salient points:

(1) A complete washability study of Pocahontas No. 3 and No. 4 seams of coal in southern West Virginia is presented, showing the difference in the characteristics of each seam. The observation is made that the No. 4 Pocahontas seam usually contains more impurities than does No. 3 Pocahontas seam.

(2) The author presents a logical method of handling the Pocahontas coals when making a two-product or a three-product separation. Due to the bony character of the Pocahontas coals it is more logical to make a three-product separation and use the intermediate product for steam purposes.

(3) Due to the friability of the Pocahontas coals the author calls attention to the degradation through the cleaning plant, and compares total preparation loss with anthracite practice.

(4) Finally, the author sets up standards of performance for mechanical preparation plants in the Southern Field and urges against the use of "rule of thumb" or visual inspection for determining the quality of the market product.

Progress Report of Committee on Evaluation of Coal for Cokemaking Purposes

By F. A. JORDAN, Youngstown, Ohio

(Tech. Pub. No. 336; Class F, Coal and Coke, No. 35)

THIS REPORT, which is accompanied by the important discussion that followed it, should be read in full, as it is impossible to do justice to it in an abstract. The principal factors it takes up are geographical considerations, market price, and ash content. The suggestion is made of a Pittsburgh seam base. The discussion brings out many important additional points.

R. H. Sweetser thought that basing valuation on ash, something not wanted, instead of on the useful carbon content, is a drawback. W. H. Blauvelt called attention to the many factors that have to be taken into account. C. E. Lesher spoke of the factor of size value and urged the necessity of taking sulfur reduction into account. J. B. Morrow discussed the degree of cleaning and its relation to fineness of crushing. J. R. Campbell, vice-chairman of the committee, spoke from the standpoint

of the steel-plant man. J. E. Little thought the suggested procedure wrong, and in reply Mr. Eavenson said: "I think I can answer your point. The committee wanted to eliminate as many of the variables as possible by confining the present problem to one coal and one particular locality. I think you have in mind taking a number of different coals from different localities at a point where these are available. In order to reach any conclusion at all, the committee decided to localize the question by confining it to the Pittsburgh seam in the Pittsburgh district. The differences in the coal here are only such differences as are found in the same seams from different mines. It makes no difference whether or not the coal is cleaned; if a mine happened to have a 7.5 per cent ash Pittsburgh coal, that was all right, and the mine that did not have it, as Mr. Robertson brought out, would have to do something to make its coal that good."

Evaluation of Coal for Blast-furnace Coke

By J. R. Campbell, Pittsburgh, Pa.

(Preprint; Class F, Coal and Coke. 1950 words)

The author has concisely brought the literature on the subject up to date. A new factor brought out within recent years, in addition to the ash evaluation of coke in blast-furnace practice, is the importance of the heavy-gravity material in coal used for cokemaking purposes and its effect on coke structure. The modern conception among the best thought in the industry is that the fine sizes of coal should be cleaned efficiently in order to eliminate the innumerable cross fractures in coke caused by their presence.

Coal Classification; a Review and Forecast

By George H. Ashley, Harrisburg, Pa.

(Trans., Coal Div., 512. 2200 words)

The writer reviewed recent coal-classification history pointing out that his proposed scheme presented in 1919 was unique in: (1) including "moisture"; (2) giving major emphasis to physical properties; (3) basing minor divisions on B.t.u. values; (4) in recognizing 36 classes; (5) in proposing mineral names; (6) and (7) and a letter code that covered grade as well as rank of coals. As a result of 10 years' further study the writer concludes that the percentage of fixed carbon in the ash-free, proximate analyses, including "moisture," offers the best basis for classifying coals by rank.

Outline of a Suggested Classification of Coals

By David White, Washington, D. C.

(Trans., Coal Div., 517. 4500 words)

This paper outlines the principal groups or divisions of a classification of coals based on the evolution of the organic deposit from peat to graphite. The different types of coal, such as common humic and cannels,

run parallel in the ascending order of the proposed series of groups or subdivisions, and precedence is, so far as possible, given to physical characters in the differentiation of these groups or major divisions.

Natural Groups of Coal and Allied Fuels

By Marius R. Campbell, Washington, D. C.
(Trans., Coal Div., 489. 6500 words)

The writer proposes to divide all coals into four great groups: (1) Browncoal group: *A*, peat class; *B*, German brown coal; *C*, lignite class. (2) Hydrobituminous group: *A*, high-moisture class; *B*, low-moisture class. Bituminous group: *A*, high-volatile class; *B*, low-volatile class. Anthracite group: *A*, high-volatile class; *B*, low-volatile class; *C*, meta-anthracite class.

Status of Coal Classification in Canada

By R. E. Gilmore, Ottawa, Ont.
(Trans., Coal Div., 529. 5400 words)

Review of material previously supplied to the Canadian and American Coal Classification Committees. The reserves, production and general uses of Canadian coals are summarized and a review of the classification schemes advanced for Canadian coals is given. The classification for Canadian "Customs purposes" is outlined, with comments on present nomenclature. The organization and activities of the Canadian Coal Classification Committee are also reviewed. Special attention is called to the coal classification work of Stansfield, with particular reference to the coals of Alberta. Arbitrary dividing lines between the duty-free anthracites and the dutiable bituminous class are discussed, and that between the bituminous coals and the lower rank noncoking "lignitic" coals is explained.

Multibasic Coal Charts

By Harold J. Rose, Pittsburgh, Pa.
(Trans., Coal Div., 541. 5800 words)

Multibasic coal charts present many advantages for the systematic study of coal analyses and physical properties. By using this form of chart, it is possible to accurately graph and present for simultaneous comparison, fixed carbon, volatile matter, calorific value, carbon, hydrogen and various physical properties of coal, on any moisture and purity basis. The degree of correlation existing between the different chemical and physical properties is plainly shown. It is also possible to show on these charts the ratios such as hydrogen: oxygen, carbon: hydrogen, etc., which have been proposed as indices for the evaluation of coal. Similarly the locus of the Goutal formula for calculating calorific value from proximate analysis, and other formulas, can be shown. The possible value and limitations of such ratios and formulas can be more satisfactorily understood from the multibasic coal chart, than in any other way.

These charts also possess unique value for comparing the various coal classification systems that have been proposed. Examples are given of 40 different coals ranging in rank from peat to anthracite, and classified according to six different systems, on various moisture and purity bases.

Changes in Properties of Coking Coals Due to Moderate Oxidation during Storage

By H. J. Rose and J. J. S. Sebastian, Pittsburgh, Pa.

(Trans., Coal Div., 556. 12,000 words)

Samples of freshly mined, pulverized Pocahontas, Powellton, and Elkhorn seam coals were oxidized under laboratory conditions designed to produce changes similar to those occurring during coal storage. The various chemical and physical properties of the coals showed progressive and characteristic changes due to oxidation. After the samples had lost all of their agglutinating power (by the Marshall-Bird method), the proximate and ultimate analyses and calorific values were still within 4 per cent of their original values. Changes of this magnitude can, of course, be readily demonstrated by ordinary analysis methods. However, the value of coal for practical coking purposes is affected by oxidation long before its agglutinating power is completely destroyed. Allowance must also be made for variations in analytical values caused by imperfections of sampling and analysis. It therefore appears that slight oxidation may affect the practical coking value of coal before changes in composition can be detected with certainty.

It has been found that the agglutinating property is very sensitive to moderate oxidation, and is subject to much greater change than any other property that was examined. The agglutinating test can be used to estimate the degree of change which has occurred during the storage of coking coals, provided that the characteristic behavior of each coal undergoing test is known. Broadly speaking, a large increase in oxygen content causes a decrease in agglutinating value, but the close relation that has sometimes been predicted does not hold true for all coals. The agglutinating value may even increase distinctly during the early stages of oxidation. No attempt is made to correlate agglutinating power with practical coking properties, nor to compare the Marshall-Bird test with similar methods.

The color intensity of the alkaline extract from coal increases with oxidation, but no correlation was found between this property and the agglutinating value.

Review of Methods Used in Coal Analysis, with Particular Reference to Classification of Coal

By A. C. Fieldner, Washington, D. C.

(Trans., Coal Div., 585. 5400 words)

Methods used for sampling and analyzing coal are reviewed from the time of the first published American analyses to the present time and the influence of variations in methods on these results is discussed and evalu-

ated. The normal variations in analytical results by the present American standard method are given and it is recommended that the committee consider the methods of the American Society for Testing Materials as standard for coal classification.

Present Status of Ash Corrections in Coal Analysis

By A. C. Fieldner and W. A. Selvig, Washington, D. C.
(Trans., Coal Div., 597. 6300 words).

For purposes of coal classification it is desirable to know the composition and calorific value of the coal substance free from ash-forming minerals. Either the coal may be freed from most of the ash-forming material before analysis by float-and-sink or acid-treating methods or the analysis may be calculated ash-free, using an ash value corrected back to the weight of the original mineral matter in the coal. The paper reviews the methods suggested by Parr and by others for correcting ash and presents experimental data from the Bureau of Mines laboratories on the amounts of water of composition, carbonate carbon and organic sulfur in various coals and shales. Correction formulas are tested by float-and-sink separations of coal and impurities. It is concluded that in general the Parr formula gives more concordant unit values than simple moisture-and-ash-free computation, but notable exceptions occur with all calculated results. The authors are doubtful whether any system of coal classification can be devised which will draw such fine distinctions as to require a corrected ash value.

Determination of Mineral Matter in Coal and Fractionation Studies of Coal

By E. Stansfield and J. W. Sutherland, Edmonton, Alta.
(Trans., Coal Div., 614. 4200 words)

This paper describes an experimental method for determining the relation between the mineral matter present in coal and the ash left when the coal is burned, as well as the calorific value of the pure coal. A centrifuge cup is shown by which a coal sample can be readily fractionated with heavy solutions for the above determinations. Calorific value-ash curves are shown and their significance discussed; also the errors involved in the usual calculations of the calorific value of ash-free coal are demonstrated. The same method of study is applied to the measurements of oxidation or weathering of coal.

Splint Coal

By Reinhardt Thiessen, Pittsburgh, Pa.
(Trans., Coal Div., 644. 11,000 words)

Splint coals, *Mattkohle* or *Streifenkohle* or durit, and hards or dulls or durain represent groups of coals that may be put under one type and should preferably be classed under one name. They are present in bands varying from a fraction of an inch to several feet in thickness, inter-

layered in a bed of ordinary humus coals, or they may constitute the whole bed. Splint coals are radically different from ordinary coals and are easily distinguishable from them. They are of an irregular lumpy nature, with an irregular rough fracture, and are rarely smooth. They are dull, grayish black in color. They have a granular consistency, are very hard, solid and tough, and produce very little fines in breaking up. Splint coals are of a relatively higher specific weight, of a different chemical composition and behavior on chemical treatment, and are slightly higher in volatile matter than the ordinary humus coals of the same rank. The average ash content of splint coals varies but the average is relatively high—higher than that of pure anthraxylon and lower than that of fusain; the ash is light gray, almost white, in color, and consists mainly of aluminum silicate. Splint coals are generally considered to be non-coking. They are composed largely of a dull granular attritus, intercalated with numerous thin sheets of anthraxylon.

Commercial Classifications of Coal

By F. R. WADLEIGH, New York, N. Y.

(Trans., Coal Div., 673. 12,000 words)

THE WRITER describes the bases of the various classifications of coal in commercial use today in the United States, and in order to illustrate the use of these different bases, presents: (1) Table of market descriptions as listed for price quotations; (2) Tidewater Coal Exchanges Classification by pools; (3) group classification of U. S. eastern coals available for export; (4) Ore and Coal Exchange Classification of coals at Lake Erie ports for water shipment; (5) commercial classifications of British, German, Belgian and French coals.

Properties of Coal Which Affect Its Use for the Manufacture of Coal Gas, Water Gas, and Producer Gas

By GILBERT FRANCKLYN, New York, N. Y.

(Trans., Coal Div., 706. 3100 words)

THIS paper was a subcommittee report to one of the Committees appointed by the American Bureau of Engineering Standards on Coal Classification. It gives a brief summary of the processes of manufacture of the three gases mentioned, and lists the characteristics of the coals suitable for each process. It also shows the regulations in force in different states on the quantity of sulfur allowable in gas manufactured for public distribution.

Commercial Description of Pennsylvania Anthracite

By E. W. PARKER, Philadelphia, Pa.

(Trans., Coal Div., 699. 3000 words)

ANTHRACITE, as sent to market, comes under three general terms of description: characteristics, source and size. It is generally classified as white ash, red ash, or Lykens Valley. The white-ash coals are sometimes further described as "hard" or "free-burning." Lehigh white ash

would come under the former description and Shamokin white ash under the latter. White-ash coal is produced in all parts of the anthracite region. It is supplied by every characteristic bed beginning with the C, Gamma or Five-foot (the third bed counting up from the conglomerate) up to and including the Holmes.

In fracture anthracite ranges from flaky and shelly up to nearly cubical. Conchoidal fracture predominates. Its color ranges from clear, shiny black, into bluish and grayish tinges, with or without luster. Anthracite gives a black streak.

Commercially, anthracite is classified with respect to sizes. Standard testing screens for the domestic sizes are as follows, all meshes being round: Broken, 4-7/16 in. through, 3-7/16 in. over; egg, 3-7/16 in. through, 2-8/16 in. over; stove, 2-8/16 in. through, 1-9/16 in. over; chestnut, 1-9/16 in. through, 11/16 in. over; pea, 12/16 in. through, 8/16 in. over.

PETROLEUM AND GAS

Petroleum Industry in 1929

(Min. & Met., January, 13. 2500 words)

Introduction and Summary, by Joseph B. Umpleby, Oklahoma City, Okla.

Production, by C. P. Watson, Fort Worth, Tex.

Production Engineering, by C. V. Millikan, Tulsa, Okla.

Refinery Progress, by A. D. David, New York, N. Y.

Petroleum Economics, by W. A. Sinsheimer, New York, N. Y.

Report of Committee on Unit Operation of Oil Pools

(Trans., Petrol. Devel. & Tech., 11. 2500 words)

General summary by the Committee which outlines the progress of unit operation of oil pools in United States and foreign fields, classifies oil and gas pools for the purposes of the study of unit operation and discusses the future scope of the A.I.M.E. unit operation study.

Unit Operations in Eastern United States and in Foreign Countries

By H. H. Hill and E. L. Estabrook, New York, N. Y.

(Trans., Petrol. Devel. & Tech., 17. 2900 words)

This report summarizes the information that was obtained by the Committee on Unit Operation in Eastern United States and in the foreign countries.

Unitized Operations in Oklahoma and Kansas

By A. W. Ambrose and C. E. Beecher, Bartlesville, Okla.

(Trans., Petrol. Devel. & Tech., 24. 4500 words)

This paper summarizes data on unitization projects in Oklahoma and Kansas as obtained from replies to questionnaires sent out by the

A. I. M. E. committee for these states. The reports received covered a total of 171,420 acres in Kansas and 49,350 acres in Oklahoma.

Unit Operation and Unitization in Arkansas, Louisiana, Texas and New Mexico

By F. H. Lahee, Dallas, Texas

(Trans., Petrol. Devel. & Tech., 34. 4000 words)

In Arkansas two unit operations were reported; in Lousiana, none; in Texas, at least eight; and in New Mexico none. All the unit operations actually put into effect were conducted successfully, but several, and perhaps many, attempts at unitizing other blocks failed. The general consensus of opinion is that, in theory, the advantages of unit operation are unquestioned. The most frequent objections to it have been raised by the smaller operators who are jealously guarding their independence from feared encroachment by the major companies. Their attitude is easily understood but is not wholly warranted.

The most important unit operation now in progress in the four states mentioned is that at Van, in Van Zandt County, Texas, where five companies participate in development costs and profits on a block of nearly 6,000 acres. The outcome of this operation will be watched with interest. Several significant proration plans are in effect, notably in the Yates, Hendricks, Chalk-Roberts, and Darst Creek pools, all in Texas.

Study of Unitization in the Rocky Mountain Region

By F. E. Wood, Casper, Wyoming

(Trans., Petrol. Devel. & Tech., 43. 2700 words)

The Rocky Mountain region enjoyed the fruits of unit operation at an early date and it is concluded from this unitization study that there is an appreciable saving in both development and production costs as well as the elimination of waste of a natural resource. The fields are classified into four types—Unit Operation, Near-unit Operation, Cooperative Agreements, and Undeveloped Fields—and data by fields presented for each type.

Unit Operation in Salt Creek Field

By Rocky Mountain A. I. M. E. Unitization Committee

(Trans., Petrol. Devel. & Tech., 48. 1300 words)

This is a brief account of the history of unit operation in the Salt Creek field, Wyoming, from the time of the agreement to prorate production in March, 1921, to February, 1930. Savings in development and production costs resulting from the unit plans are estimated at $11,650,000.

Unit Operation in the Rock River Field, Wyoming, with Notes on the Rex Lake Field, Wyoming

By Wilson B. Emery, Casper, Wyoming
(Trans., Petrol. Devel. & Tech., 51. 3000 words)

Unit operation in the Rock River field has resulted in economies running into millions of dollars in value. Drilling cost would have been approximately 100 per cent more and producing expense over the life of the field 70 per cent more than it would have been under unit operation while general and camp expense would have been increased perhaps one-third. Orderly development, balancing production with market requirements during the period of flush production, conservation of gas and utilization of excess gas for repressuring are other noteworthy benefits of unit operation in this field. At Rex Lake the saving through unit operation is estimated at close to $500,000.

Unit Operation as Proposed for the Hiawatha, South Baxter Basin and North Baxter Basin Gas Fields in Southwest Wyoming and Northwest Colorado

By William T. Nightingale, Rock Springs, Wyoming
(Trans., Petrol. Devel. & Tech., 57. 3500 words)

The writer describes the three fields and discusses the prospects for production under a unit operation plan. He concludes that unitization appears highly justifiable on the basis of economy and conservation.

Unit Operation in Hidden Dome Gas Field, Wyoming

By Wilson B. Emery, Casper, Wyoming
(Trans., Petrol. Devel. & Tech., 66. 1300 words)

Investment and expense in the Hidden Dome gas field have been held down to a minimum by unit operation, and the gas conserved and used as fuel. It is doubtful whether these results would have been even closely approached under diversified ownership and management.

Unit Operation in California

By Joseph Jensen, Los Angeles, Cal.
(Trans., Petrol. Devel. & Tech., 69. 4600 words)

No outstanding example of an important producing unit operation exists today in California. What has been accomplished may be classed as near-unit operation. In these near-unit fields controlled deeper drilling is still possible, and in all an excellent opportunity for gas storage and repressuring. An excellent practice in some of these near-unit fields has been the limitation of penetration. Unit plans have been made for wildcat areas. No better example of controlled drilling exists in California than the Ventura Avenue field, which has been developed slowly by regulated drilling since 1922. There still remains from two to five years' drilling in this field. Some plan of unit operation or regulated

development is absolutely necessary in the Kettleman Hills field, and this may reasonably be expected. Unit operation is being emphasized, as new discoveries demonstrate the need of control of excess production. It is well to point out that unit operation will pay its way in lessened drilling and producing costs and in added oil recovery.

Unit Operation Proposal for Kettleman Hills Field

By Robert C. Patterson, Taft, Cal.

(Preprint; Class G, Petroleum and Gas. 2700 words)

OUTLINE of agreements, committee work and plans, production of the field and probable results of restricted operation.

For discussion (6000 words) by T. E. Swigart, see Min. & Met., December. If operators at Kettleman drill without regard for market demands, at least 300,000 bbl. per day will be developed and the excess over California's gasoline requirements will have either to go to storage or be shipped to compete in the markets of eastern United States and Canada. If Kettleman is allowed to produce its present potential of 90,000 bbl. per day of oil, it is estimated that approximately 1,370,000,000 cu. ft. of gas per day would have to be wasted. After study it has been decided that unit operation with a pooling of production and a calculated share for the production, but would permit obtaining the field's production from a few wells as compared to 100 or so that would be required under competitive conditions.

Control of California Oil Curtailment

By Robert E. Allen, Los Angeles, Cal.

(Preprint; Class G, Petroleum and Gas. 9500 words)

THIS paper outlines the history and practice of oil proration as developed and applied in California. Reference is made to the economic aspects and advantages of balanced oil production and many of the basic principles of organized curtailment are explained.

Details of engineering control are given and illustrated graphically. Some new principles relating to oil production are mentioned and especial attention is directed to the fact that proration has resulted in a greatly increased knowledge of the mechanics of oil production and of the factors affecting both ultimate recovery and rate of recovery.

Principles of Unit Operation

By Earl Oliver, Ponca City, Okla., and J. B. Umpleby, Oklahoma City, Okla.

(Trans., Petrol. Devel. & Tech., 105. 5500 words)

THIS paper reviews the simple principles of unit operation and was submitted by the writers as of possible helpfulness in connection with the study of unit operation by the Petroleum Division. It makes brief reference to the changing conditions facing the United States petroleum

industry and discusses the disadvantages of competitive extraction and the problems surrounding the general adoption of unit operation. The advantages of and the objections to unit operation are listed and there are also presented a number of observations which the writers believe reflect sentiment on the movement for unit operation.

Economic Aspects of Unit Operation of Oil Pools

By Joseph E. Pogue, New York, N. Y.

(Min. & Met., November, 540. 2600 words)

The conclusions drawn are as follows:

Unit operation is developing as a superior economic method of producing crude petroleum. Unit operation offers the means for reducing costs, eliminating waste, and gaining the economic advantage of a *reserve without a potential.* The progress of unitization is being facilitated by the principle of differential cost, the principle of differential extraction, the drift toward large-scale operations, the need for rationalization of supply, and the public interest in conservation. Progress in unitization is opposed by custom and the archaic law of oil and gas, which may be subject to change. Unitization offers the most promising means available for placing the production of crude petroleum on a sound economic basis.

Some Development and Operating Economies of Unit Operation

By Sam Harlan, Bartlesville, Okla.

(Trans., Petrol. Devel. & Tech., 118. 4500 words)

An investigation based on actual figures revealing the practical advantages of unitization in so far as the expenses of development, operation and maintenance are concerned. It is shown that a total ultimate saving equivalent to 46 per cent of the total actual development cost might be anticipated solely from the normal benefits to be derived from the single management of a group of leases comprising a pool.

Production Engineering in 1929

By C. V. Millikan, Tulsa, Okla.

(Trans., Petrol. Devel. & Tech., 142. 1800 words)

Production engineering continued its rapid progress during 1929. Deviation of drill holes received much attention. Deep wells with high initial oil and gas production and closed-in pressure in excess of 2000 lb. were common and required larger and stronger equipment for drilling and producing. Pumping equipment was studied closely, especially sucker rods. Underground storage of crude oil was practiced by a California operator.

Equilateral Triangular System of Well Spacing

By C. S. Corbett, New York, N. Y.

(Trans., Petrol. Devel. & Tech., 168. 2700 words)

THE orientation of the well-spacing coordinate system with respect to the trend of the structure and the steepness of dip is discussed and data are presented to show the ratio of formation distance between wells for two orientations and for various angles of dip. It is shown how one-third or one-fourth of all the locations for which the coordinate system provides may be drilled and yet maintain equilateral triangular spacing. For repressuring a depleted field, the use of one-third of the wells for gas input, chosen according to the arrangement indicated for maintaining equilateral triangular spacing, should be more effective than using one-half of the wells according to the "five-spot" plan.

Density of Oil-gas Columns from Well Data

By William Victor Vietti, Wink, Texas

(Min. & Met., October, 453. 1900 words)

CURVES are constructed from well data by plotting oil and gas production against casinghead pressures. The extension of the curve to zero pressure determines the open flow potential of the well. Extension of the curve to zero production determines the closed-in pressure with the fluid disseminated as during actual flowing conditions. Subtraction of this pressure from the bottom-hole pressure gives the pressure exerted by the column at zero flow. Data from several wells in the Yates field allow the construction of curves showing the change in density at zero flow as the amount of gas increases.

Some Principles Governing the Choice of Length and Diameter of Tubing in Oil Wells

By J. Versluys, The Hague, Netherlands

(Tech. Pub. No. 344; Class G, Petroleum and Gas, No. 31)

THIS is a continuation of the discussion which the author initiated in Tech. Pub. No. 213, where he set forth the theory of flowing oil wells, and in it he analyzes the mathematical principles involved in selecting the length and diameter of oil-well tubing for a given condition of flow. From its nature it is impossible to make a summary of it and everyone interested should read the original paper.

Velocity of Flow through Tubing

By E. L. Davis, Los Angeles, Cal.

(Preprint; Class G, Petroleum and Gas. 2400 words)

CONTINUOUS flow of an oil well depends on maintaining oil and gas in the flow tube in an intimate mixture. The maintenance of this mixture is dependent on velocity. Minimum velocity at the base of the flow tube, of from 4 to 7 ft. per second, is found necessary to prevent oil from

separating out of the mixture and causing heading or cessation of flow. Greatest oil production occurs with velocities of from 9 to 18 ft. per second at the base of the flow tube. A nomographic chart permits quick calculation of velocity of oil-gas mixture at any point where the pressure is known.

Increasing the Ultimate Recovery of Oil

By S. F. SHAW, Tulsa, Okla.

(Tech. Pub. No. 358; Class G, Petroleum and Gas, No. 35)

THE laws governing the filtering effect of mixtures of oil and gas in passing through a porous sand medium probably have far greater effect on production of oil, under the various conditions that exist in the sand, than the single factor of viscosity as influenced by back-pressures. In other words, viscosity is only one factor under existing underground conditions, and probably is a relatively unimportant factor. The explanation suggested by the author is that while a high back-pressure may reduce the viscosity of the oil, it tends to reduce the velocity of flow through the sand to a point where the moving gas will not carry with it the maximum load of oil. In other words, the available energy in the gas is not utilized to maximum advantage. The method of production tentatively suggested for obtaining the maximum ultimate recovery of oil is to enlarge the filtering area through the oil sand at the bottom of the well as much as practicable and to reduce back-pressures at the sand to the critical differential pressure.

Effect of Proration on Decline, Potential and Ultimate Production of Oil Wells

By H. H. POWER and C. H. PISHNY, Tulsa, Okla.

(Preprint; Class G, Petroleum and Gas. 6600 words)

A STUDY of production curves on various pay horizons shows that in many fields wells of high structural position eventually become edge wells because of encroaching water. Continuous production under pressure control rather than the so-called intermitting might delay water encroachment, although intermitting has actually increased the potential and daily production of certain wells having low fluid levels. On one lease the use of gas-lift retarded the water-flood line, while pumping wells of similar structural position reached their economic limit much sooner. In many cases in fields operated under proration agreements, deferred production will be recovered. The ultimate recovery of some wells, especially edge wells, will be decreased, but a close study of field conditions and operating methods should materially reduce such losses. Proration affords opportunity for study of operating methods and encourages experimentation in pressure control and water control, leading to more efficient operation and greater ultimate recovery.

The Effect of Edge Water on the Recovery of Oil

By H. H. Wright, Tulsa, Okla.

(Tech. Pub. No. 367; Class G, Petroleum, No. 36)

Encroaching edge water influences the recovery of oil to an extent greater than is commonly realized. In many fields, particularly in the Mid-Continent, edge water affects the rate of decline even in the flush stage of production. Many wells from which the production is well sustained over many months or years owe their high recovery to natural water flooding of the producing sand. In this paper it is shown that the rate of water encroachment is one of the factors determining the efficiency of natural edge water flooding. Pressure head on the water, sand permeability or resistance to flow, and rate of depletion of oil and gas content are indicated as the variables mainly controlling the rate of water encroachment. It is pointed out, for a given water head and sand permeability, that regulation of the third variable should control encroachment. The value of this variable depends on well spacing, rate, method and sequence of drilling, and production methods. In order to make this control most effective it must be exerted under some system whereby the operating unit shall consist of the whole pool or structure rather than of the individually owned and operated lease.

Recent Developments in Flooding Practice in the Bradford and Richburg Oil Fields

By Charles R. Fettke, Pittsburgh, Pa.

(Tech. Pub. No. 328; Class G, Petroleum, No. 30)

The price of oil in this field for the first six months of 1929 was the highest since 1924, with the result that more new development work was undertaken and completed. Improvements were made in the "five-spot" method of working and these are discussed in some detail, as well as several others which have been employed. The latter part of the paper discusses the present status of air and gas drives.

Modern Practice in Water-flooding of Oil Sands in the Bradford and Allegany Fields

By Paul D. Torrey, Bradford, Pa.

(Trans., Petrol. Devel. & Tech., 259. 7500 words)

The efficiency of water-flooding operations in the Bradford and Allegany fields has been constantly improved during the past 10 years. Economic factors have been most favorable for expansion during this time. Improvements in operating technology that are noteworthy consist of filtering and treating the water used in flooding, the use of central water-pumping plants, the extensive coring of the oil sand and the installation of Diesel engines for power. Much knowledge has been gained regarding the factors determining the most advantageous well spacing to use. It has been found that the porosity, permeability and oil content of the sand

are very important factors and must be considered. The application of the delayed system of drilling oil wells has yielded very satisfactory results.

Encroachment of Waters at Santa Fe Springs

By DONALD K. WEAVER, Los Angeles, Cal.

(Min. & Met., October, 472. 2500 words)

EIGHT different oil zones have been identified and produced at Santa Fe Springs, of which three or four are in turn divided into two or three parts.

Each of these oil zones has had waters of varying salinity associated with it, either within or at the bottom of the zone. The most prolific zones of the field, to date, are the Meyer and O'Connell zones. Also there are more definite data on position and rates of water encroachment in these zones, on account of the large number of wells that were left to produce in them. For this reason, waters in these two zones are discussed in detail.

Water Invasion—McKittrick Oil Field—an Apparent Reversal of Normal Oil Field History

By JOSEPH JENSEN, Los Angeles, Cal., and J. B. STEVENS, Fellows, Cal.

(Min. & Met., October, 470. 2000 words)

MCKITTRICK oil field, Kern County, California, having produced over 94,000 bbl. per acre, has had interesting production history. Water appearing early rapidly increased until for five years the field was producing 1,000,000 bbl. of water and 200,000 bbl. of oil per month. For the last 10 years the water volume has decreased and fluid table has dropped over 200 ft. These changes are due in part to draining for field and domestic use of water from near-by higher basin. Thus the higher basin ceased to be a source of surface water and continued pumping of the wells lowered their fluid level. Invasion of bottom water is limited in volume by faulting, porosity or low head. During these changes, oil production remained unaffected, the decline curve being remarkably flat. Remaining history will probably be marked by normal oil decline, with water problem secondary.

Water Encroachment in the Salt Creek Field

By EDWARD A. SWEDENBORG, Midwest, Wyoming

(Min. & Met., July, 341. 3500 words)

THIS paper gives the results of a careful study in which a distinction is made between condensed water from gases and that which is invading the sand.

Valuation of Flood Oil Properties

By EUGENE A. STEPHENSON and I. G. GRETTUM, Pittsburgh, Pa.

(Tech. Pub. No. 323; Class G, Petroleum and Gas, No. 28)

FLOODING operations in the Bradford field have been carried on so long that this area is now the outstanding illustration of oil recovery by

water drive. Although originally the flooding process was largely the result of accidents to casing and tubing it has gradually passed from the accidental to the engineering stage and is attracting the attention of the most efficient technical organizations.

Old methods of flooding which succeeded in raising the oil production from 1/8 bbl. per well per day to a maximum of 2 to 3 bbl. per well per day have been superseded by other types of floods, which have raised the production to as much as 80 bbl. per day. The ratio of oil lifted to water lifted has also, under the most efficient operations, been greatly reduced. Technical details of flooding operations are described and the various factors involved in the estimation of the recovery per acre, the rate of production, the cost of operations and finally the present worth of a property at which it is proposed to conduct flooding operations.

Mechanics of a California Production Curve

By STANLEY C. HEROLD, Los Angeles, Cal.

(Tech. Pub. No. 322; Class G, Petroleum and Gas, No. 27)

DIFFICULTIES in analyzing and comparing the performance of oil reservoirs in various parts of the world do not rest solely upon structural features and property lines. We do not complete the list of causes for these difficulties if we add lithologic features and correlation of strata encountered by the drill, with the magnitude of pressures exerted by fluids within any of these strata. A further cause, one which the author believes will complete the list, is one involving the theoretical mechanics of fluids within porous formations—porous in the nature of interstices in sandstones and some limestones, networks of fractures in shales and cavities in otherwise compact limestones.

The confusion of ideas concerning reservoir mechanics still with us today seems largely due to our failure to recognize among all wells three great divisions of theoretical mechanics, three great systems which are distinct in themselves, and according to which each well, or better, each group of wells, must be analyzed separately. These systems have been named by the author Hydraulic Control, Volumetric Control, and Capillary Control respectively. Wells in these controls behave differently, and they must be handled differently to get the best results in production practice. The author discusses these various factors in relation to typical California wells and draws sundry conclusions.

Recent Studies of the Recovery of Oil from Sands

By JOSEPH CHALMERS, Bartlesville, Okla.

(Trans., Petrol. Devel. & Tech., 322. 2700 words)

FLOW-TUBE experiments conducted by the U. S. Bureau of Mines at the Petroleum Experiment Station, Bartlesville, Okla., to obtain data pertaining to the relative merits of various gases as pressure media in the recovery of oil from sands by the "gas drive" are described. Propane, a mixture of propane and "dry" natural gas, "dry" natural gas, air and helium were used with crude oil from the Bartlesville sand. In

addition to benefits attributed to the density and solubility of the gases, it was found that control of the rate of gas injection and a complete utilization of potential energy in the form of dissolved gas were means of attaining higher efficiencies based upon gas-oil ratio and energy expended.

Law of Flow for the Passage of a Gas-free Liquid through a Spherical-grain Sand

By WILLIAM SCHRIEVER, Norman, Okla.

(Trans., Petrol. Devel. & Tech., 329. 3600 words)

A MINERAL oil (Nujol) was passed through four different glass-sphere sands (dia. 0.025 to 0.1 cm.) having porosities varying from 35 to 39 per cent. The pressure drops through the sands varied from 0.2 to 0.8 cm. of mercury per centimeter length of sand column for the coarsest sand, and from 0.9 to 4.5 cm. of mercury per centimeter of column for the finest sand. The data showed that the rate of flow was directly proportional to the pressure drop, the diameter of sand grain raised to the 1.68 power, and the porosity raised to the power $(4.14 + 0.0141/d)$ (where d is the diameter of grain in centimeters) and inversely proportional to the length of sand column.

Variation of Pressure Gradient with Distance of Rectilinear Flow of Gas-saturated Oil and Unsaturated Oil through Unconsolidated Sands

PART I OF FINAL REPORT OF A.P.I. PROJECT NO. 33

By W. F. CLOUD, Norman, Okla.

(Trans., Petrol. Devel. & Tech., 337. 4500 words)

THIS paper contains results obtained in an investigation on the Effect of Natural Gas on the Viscosity, Surface Tension, Adhesion and General Extractability of Crude Oil, listed as Project No. 33 of American Petroleum Institute Research. The data and information compiled under Part I are the results of experiments performed in the petroleum engineering laboratory of the University of Oklahoma under the supervision of the writer.

Behavior of Gas Bubbles in Capillary Spaces

By IONEL I. GARDESCU, Pittsburgh, Pa.

(Tech. Pub. No. 306; Class G, Petroleum Division, No. 25)

THE Jamin action is the resistance offered by detached gas and liquid bubbles confined to capillary tubes by a boundary condition which develops whenever the liquid does not wet the solid walls of the capillary. The resistance opposed by the bubbles is proportionate to the variation of the solid-surface-tension at the two extremities of each individual liquid bubble and is inversely proportionate to the radius of the capil-

lary tube. For oil and sand the Jamin effect is zero or very small. The resistance p offered by a gas bubble when forced into a capillary opening is given by the equation:

$$p = 2\,S\left(\frac{1}{r_1} - \frac{1}{r_2}\right)$$

in which S is the surface tension, r_1 the interporous opening through which the bubble is forced, and r_2 the maximum curvature of the distorted bubble, when the curvature at the other extremity of the bubble is r_1. To be forced through a capillary opening of 0.011 cm. diam., a gas bubble, present in a liquid medium of surface tension equal to 23, will require a pressure zero to 4000 dynes, depending on its size. The magnitude of resistance offered by gas bubbles forced through small openings so far exceeds any possible resistance caused by the Jamin effect that the latter phenomenon need scarcely be considered in dealing with the movement of oil and gas through their natural reservoir rocks.

Microscopic Study of California Oil-Field Emulsions

By Mahmood Abozeid, Berkeley, Cal.

(Tech. Pub. No. 345; Class G, Petroleum and Gas, No. 33)

This interesting paper is only a part of the concluding chapter of the author's thesis on the study of oil-field emulsions submitted to the University of California. After reviewing the general characteristics of emulsions and the effect of heat on them, and the effect of dilution, he reviews at some length the effect of electric charges. The phenomena noted are illustrated by 31 photomicrographs.

Cementing Problem on the Gulf Coast

By H. D. Wilde, Jr., Houston, Texas

(Trans., Petrol. Devel. & Tech., 371. 4000 words)

Work in the laboratory showed that neat cement as used in cementing wells was markedly weakened if mixed with too much water and the cement subsequently prevented from settling by agitation and contamination with drilling mud. Furthermore, contamination with mud itself weakened the cement. A petrographic examination of many cement cores obtained from wells at Sugarland and Raccoon Bend in the Texas Gulf Coast indicated that the poor cores were contaminated with drilling mud. Thereafter in the field care was taken in cementing wells to use as little water as practicable, to make the water addition uniform and to avoid contamination of the cement slurries with drilling mud as much as possible. With the improved cementing technique, the percentage of failures as indicated by the appearance of cores was reduced to an almost negligible figure.

Properties and Treatment of Rotary Mud

By HALLAN N. MARSH, Los Angeles, Cal.

(Preprint; Class G, Petroleum and Gas. 7400 words)

THE large expenditures for mud used in rotary drilling of oil wells and the serious difficulties arising from the use of improper mud dictate an intensive study of the subject. The important properties of mud are discussed under the headings of weight, fineness and consistency. The distinction between weight and consistency is emphasized; also the difference between fluids and plastics. Test methods are described. Principles of common and special methods of treatment are described, with data showing results of using admixtures, chemicals, various waters, ditches, centrifuges and vibrating screens. Application of known facts and principles is proving beneficial, but further investigation is needed.

Drilling-mud Practice in the Ventura Avenue Field

By F. W. HERTEL and E. W. EDSON, Ventura, Cal.

(Trans., Petrol. Devel. & Tech., 382. 4000 words)

THE formations in the Ventura Avenue field make very little mud and it is necessary to supply mud to the wells from a clay pit a short distance from the field. The paper explains the mixing and handling of mud fluid and types of mud used and the use of admixtures for loss of circulation, squeezing and caving formations and high gas pressures. Three methods of sand elimination are discussed. The writers believe that more attention should be directed to the mud fluid and that proper mud fluid is a great aid in successful completion of wells.

Review of Oil-field Corrosion Problems for 1929

By L. G. E. BIGNELL, Tulsa, Okla.

(Trans., Petrol. Devel. & Tech., 392. 1800 words)

THIS paper briefly reviews the progress made in 1929 in the study of corrosion problems, and the application of various metals in combating them. A short summary of the tests made with aluminum paint, foil and sheet metal for vessels handling crude oils produced with appreciable quantities of hydrogen sulfide is given. Reference is made to a lecture delivered by Albert Portevin in Paris upon the results of tests of ferrous metals and the work of Dr. Logan and Dr. Scott of the U. S. Bureau of Standards on soil-corrosion problems is also touched upon.

Petroleum Economic Review for 1929

By WARREN A. SINSHEIMER, New York, N. Y.

(Trans., Petrol. Devel. & Tech., 396. 4000 words)

THROUGHOUT 1929 remedies were sought to cure overproduction and waste. Proration and voluntary curtailment not only failed but contributed to further waste and created a false sense of security. The manufacturing of gasoline without regard for market requirements falsely stimulated demand for crude and caused an oversupply of finished

products, particularly gasoline. The marketing section of the industry made progress by its adoption of the Code of Ethics as promulgated by the Federal Trade Commission and by the formation of an Export Association.

Controlled Gasoline Supply—the Key to Oil Prosperity

By H. J. Struth, Houston, Texas

(Trans., Petrol. Devel. & Tech., 408. 4500 words)

The effect of overproduction of gasoline upon the financial structure of the petroleum industry is given particular stress in this paper which emphasizes the need for rigid control of refinery still runs during 1930, in order to complete the movement toward stabilization that has apparently made itself manifest in the producing branch of the industry. On the basis of an expected gasoline demand during 1930 of 472,000,000 bbl., the writer points out that little, if any, more crude will be required than in the preceding year. The folly of running crude to stills in excess of normal requirements for gasoline and other petroleum products is illustrated by the fact that such action during 1929 cost the industry, roughly, $105,000,000; representing the difference between the actual value received from the sale of motor gasoline and the value that might have been received on the basis of prices obtaining during 1928. This and other vital data on the refining situation are graphically illustrated with seven charts, supplemented by four tables.

Economic Trend of the Petroleum Situation

By Joseph E. Pogue, New York, N. Y.

(Trans., Petrol. Devel. & Tech., 405. 1000 words)

New economic forces are at work in the petroleum industry. For several decades prior to 1911, the outstanding feature was a trend toward integration. The dissolution of the Standard Oil Co., in 1911, ushered in the second era of the petroleum industry, a period of intense competition. The industry is now entering its third stage, which is marked by a return toward integration and control, as opposed to disorganized competition.

An Economic Comparison of Developments in the South Field Oil-producing Region of Mexico

By Oliver B. Knight, Tampico, Mexico

(Tech. Pub. No. 343; Class G, Petroleum and Gas, No. 32)

After reviewing the development of the South Field structure of Mexico the author gives the data on production from the field and then takes up the costs of development, drilling and physical equipment. These costs are summarized in a table.

Problems of Petroleum

By J. Elmer Thomas, Fort Worth, Texas
(Trans., Petrol. Devel. & Tech., 423. 2200 words)

After reviewing the large discoveries in crude reserves made by producers this paper points out how refiners and marketers were thereby encouraged to expand their operations. The pressure on price exerted by the mounting surplus of "potential," and its consequent effect in rendering stored inventories excessive, is stressed. Control by enforcing conservation measures is expected to correct the overproduction of crude oil and later the overexpansion of marketing facilities, which latter could be improved by abandoning gasoline trade-marks.

Influence of Control in the Oil Industry upon Investment Position of Oil Securities

By Barnabas Bryan, New York, N. Y.
(Trans., Petrol. Devel. & Tech., 430. 1800 words)

The oils were in a continual bear market from 1919 to 1927. Since the fall of 1927 the trend of the oils has changed, first to a flattening or constant relation to the market and later the beginning of a rising series of highs and lows about the market averages. There can be no question of the cause for this change in the performance of the oil stocks, since its beginning in 1928 correlates exactly with the beginning of the correction of the most fundamental difficulty to be found in the history of oil—the inability to control its raw material in line with demand. This one fact has been the cause of irregular earnings by the good companies since oil was first produced. If the supply of crude oil could be kept in line with true demand, the earnings of oil companies would be regularly good from year to year.

Production Review for 1929

By C. P. Watson, Fort Worth, Texas
(Trans., Petrol. Devel. & Tech., 436. 450 words)

Oklahoma, Texas and California in 1929 yielded 84 per cent of the total United States production from 33.5 per cent of the total number of producing wells. Production showed an increase of 107,000,000 bbl. over 1928. In the states named the most significant development was the desire of operators to reach some agreement to bring production in line with demand.

Petroleum Production and Development in Kansas during 1928 and 1929

By Charles E. Straub and Anthony Folger, Wichita, Kansas
(Trans., Petrol. Devel. & Tech., 437. 13,000 words)

Kansas produced 38,150,878 bbl. of oil in 1928 as against 40,658,170 bbl. in 1929, thus retaining its rank as fourth among the oil-producing

states. The bulk of this oil was produced from eight counties: Butler, Sedgwick, Greenwood, Sumner, Cowley, Marion, Russell and Chautauqua. The most important discoveries in eastern Kansas were the Lost Springs and Hillsboro fields of Marion County, the Lamont and Patterson fields of Greenwood County, the Sluss field in Butler County and the State Home field in Cowley County. In western Kansas important new discoveries were the Valley Center and Greenwich fields of Sedgwick County, the Voshell and Ritz fields of McPherson County and the Raymond field in Rice County. Of salient importance to western Kansas has been the discovery of oil in three new producing horizons: namely, Pennsylvanian basal conglomerate of Pennsylvanian age and from the Simpson dolomite and Siliceous lime of Ordovician age.

Petroleum Developments in Oklahoma during 1929

By H. B. Goodrich, Tulsa, Okla.

(Trans., Petrol. Devel. & Tech., 466. 3600 words)

Total production was 252,229,474 bbl.; 1928, 247,500,851 bbl.; 1927, 276,022,024 bbl. The Oklahoma City pool was the most interesting field in 1929, from all points of view. The oil producers' main attention was toward curtailing overproduction down to market demand and seemed to be successful in the last two months. Nevertheless there were constructive drilling developments, leading to the future.

Petroleum Development in West Texas and Southeast New Mexico in 1929

By R. E. Rettger, San Angelo, Texas

(Trans., Petrol. Devel. & Tech., 476. 6000 words)

Oil or gas was discovered in 15 different localities but no one of these discoveries had proved of major importance. Total production of 124,211,518 bbl. was slightly greater than in 1928. The 1929 production alone amounted to about 40 per cent of the total oil produced in this area since discovery nine years before.

Development in East Texas and Along the Balcones Fault Zone, 1929

By F. E. Poulsen, Fort Worth, Texas

(Trans., Petrol. Devel. & Tech., 492. 4000 words)

The discoveries of the Van and Darst Creek fields were the outstanding developments in 1929. The first six months were very inactive. Five of the interior salt domes were drilled with discouraging results and the Boggy Creek field had fallen far short of expectations. The situation improved in the latter half of the year. The Van field, discovered in October, gained for East Texas a permanent place in point of future potential production. Wildcatting during the first half of 1929 in the Luling district was dilatory. At the end of the year, interest in the Balcones fault zone fields centered in Darst Creek, discovered in July.

Petroleum Developments in North Central and West Central Texas during 1929

By J. WHITNEY LEWIS, Fort Worth, Texas

(Trans., Petrol. Devel. & Tech., 501. 1800 words)

THERE was a marked lull in both prospecting and development. Archer, Coleman, Cooke, Wilbarger and Young were the counties most active. Total gross production was 52,611,000 bbl., 3 per cent more than for 1928.

Petroleum Development in Southwest Texas during 1929

By OLIN G. BELL, Laredo, Texas

(Trans., Petrol. Devel. & Tech., 505. 2200 words)

PRODUCTION for this area in 1929 was 6,281,096 bbl., the Refugio field producing 2,106,055 bbl. Production data by fields and wells are shown in a table and developments throughout the area are discussed in detail.

Petroleum Developments in Texas Panhandle in 1929

By WILLIAM E. HUBBARD, Amarillo, Texas

(Trans., Petrol. Devel. & Tech., 510. 2250 words)

THE Panhandle produced 31,105,600 bbl. in 1929; 24,837,800 in 1928; 41,009,500 in 1927; 26,009,300 in 1926. Production in Gray County increased from 7,846,900 bbl. in 1928 to 18,563,700 bbl. in 1929. Four years ago the Panhandle was generally looked upon as a vast reservoir of oil from which production could and would be withdrawn as the economic situation warranted. Since then little has happened to change this picture save that the average indicated ultimate production per oil well drilled has been revised upward from about 75,000 to 85,000 bbl., allowing oil to be produced profitably at a correspondingly lower market.

Petroleum Developments in Gulf Coast of Texas and Louisiana during 1929

By R. H. GOODRICH, Houston, Texas

(Trans., Petrol. Devel. & Tech., 515. 2250 words)

DEVELOPMENT was little different from that of any other year despite the somewhat depressed condition of the oil business in general. The year was marked by: (1) An intensive geophysical campaign in the search of deep-seated salt domes; (2) the rather successful exploration and development of lateral sand production on the flanks of some of the older domes. Many of the fields showed a marked decline in production; Barbers Hill production increased by 4,163,950 bbl. over 1928.

Petroleum Developments in Arkansas in 1929

By H. W. BELL, El Dorado, Arkansas

(Trans. Petrol. Devel. & Tech., 522. 1100 words)

THERE was considerable prospecting for new supplies of oil in Arkansas during the past year, regardless of the overproduction affecting the

industry throughout the country. Justification for this new work was not lacking, as the local markets more than threatened to absorb a declining production. There were, however, no important new discoveries in the state during 1929.

Petroleum Developments in California during 1929

By **Desaix B. Myers**, Los Angeles, Cal.

(Trans., Petrol. Devel. & Tech., 525. 3000 words)

THE consistent upward trend in crude oil production prevailing in California throughout the greater part of 1929, was effectively checked in November by a curtailment program instituted by mutual agreement between operators in four of the major fields. This program artificially reduced daily production to approximately the same daily figure as prevailed in December, 1928, but the large amount of deep drilling during the latter part of 1928 and during 1929 established a potential far in excess of refinery needs in the state. Low prices for crude oil which prevailed in California during 1928 continued during 1929. General curtailment continued in the older fields in San Joaquin Valley. Total production was 292,036,911 bbl., 25.88 per cent above 1928.

Development Program in a Part of the Ventura Avenue Oil Field

By **Joseph Jensen**, Los Angeles, Cal., and **F. W. Hertel**, Ventura, Cal.

(Min. & Met., October, 475. 3500 words)

THE Ventura Avenue field has only one distinctive shale body and one intermediate water between the great thickness of oil zone, leaving the lower 3300 ft. of the present known oil zone without any definite marker to divide it into different zones. In earlier years wells penetrated so much oil zone that recovery per foot of penetration was materially reduced. The differences in quantity of oil and gas production, areal extent of the zones, location of edge waters and practical drilling depth are the factors that have been used by the Associated Oil Co. in the eastern part of the field to divide this 3300 ft. of oil zone into three parts for three classes of wells. There is no longer any temptation to deepen wells when they are still producing commercial quantities of oil for larger production below. The program meets present needs and fits into a scheme for development of deeper zones at the proper time.

California Oil Production Outlook for 1930

By **H. Norton Johnson**, Los Angeles, Cal.

(Min. & Met., April, 216. 4400 words)

DURING 1929 production reached new heights and new depths. Production in 1930 depends chiefly on economics and psychology.

Petroleum Production and Development in the Rocky Mountain District in 1929

By F. F. Hintze, Salt Lake City, Utah

(Trans., Petrol. Devel. & Tech., 533. 2700 words)

Under the general policy of retrenchment in wildcat drilling and proration of production, there were fewer oil operations in the Rocky Mountain district in 1929 than for several years previous and oil production declined approximately 2,500,000 bbl. Production in the Salt Creek field declined just about this amount. Deep drilling at Salt Creek was showing up large oil reserves under that field, and with unit operation of the deeper wells there should be a large recovery and increased life for the field.

Petroleum Development in 1929 in the North Rocky Mountain Region, including Wyoming, Montana and Alberta

By Ralph Arnold, Los Angeles, Cal., and O. I. Deschon, Great Falls, Mont.

(Trans., Petrol. Devel. & Tech., 539. 2250 words)

Deep drilling was the keynote of the more important developments in the North Rocky Mountain region during 1929, with Montana recording the most important achievements through discovery of three new oil fields. Alberta produced 1,000,000 bbl. of 74° gravity naphtha, showing a considerable gain over 1928, developed a commercial oil pool in Turner Valley and extended that field 8 miles southward.

Appalachian Petroleum and Natural Gas Fields during 1929

By Charles R. Fettke, Pittsburgh, Pa.

(Trans., Petrol. Devel. & Tech., 544. 1800 words)

The outstanding event was the intensive drilling activity in the Bradford and Richburg pools of northwestern Pennsylvania and southwestern New York State, particularly the former, in connection with the continued extension of the five-spot system of water-flooding. More new development work was undertaken and completed than during any previous year since reclamation methods were first applied. This was due in part to an excellent price for oil and in part to discovery of the fact that more intensive methods (five-spotting) yielded greater profit. No new oil pools of significance were discovered.

Petroleum Development in Indiana and Illinois in 1929

By Alfred H. Bell, Urbana, Ill., and Paul F. Simpson, Indianapolis, Ind.

(Trans., Petrol. Devel. & Tech., 548. 600 words)

The year was one of continued activity but the new production obtained was not sufficient to offset the decline in the production of the old wells.

Petroleum Developments in Mississippi during 1929

By Ralph E. Grim, University, Miss.

(Trans. Petrol. Devel. & Tech., 550. 650 words)

This paper discusses structural features of the state and summarizes 1929 prospecting by districts.

World Petroleum Production during 1929

By Valentin R. Garfias, New York, N. Y.

(Trans. Petrol. Devel. & Tech., 552. 1400 words)

This paper discusses production throughout the world and is accompanied by a table showing 1929, 1928 and 1927 figures.

Petroleum Development in Venezuela during 1929

By E. L. Estabrook and J. A. Holmes, New York, N. Y.

(Trans., Petrol. Devel. & Tech., 556. 2700 words)

Venezuela continued to demonstrate that it is destined for many years to come to be one of the most important sources of crude petroleum. Total production was approximately 135,953,000 bbl., a 28 per cent increase over 1928.

Russian Oil Fields in 1928 and 1929

By Basil B. Zavoico, Tulsa, Okla.

(Trans., Petrol. Devel. & Tech., 564. 3100 words)

Considerable progress was made throughout the Russian petroleum industry in 1929. Perhaps the most remarkable achievement was the execution of the programs set out by the Central Planning Bureau of the Department of Commerce. These programs were increased twice during the operating year, necessitating drilling the richest reserves, instead of developing gradually as originally planned. The production of all Russian fields increased by 14,200,000 bbl. as compared with the 1927-1928 operating year.

Mexican Oil Fields during 1929

By Valentin R. Garfias and C. O. Isakson, New York, N. Y.

(Trans., Petrol. Devel. & Tech., 573. 2000 words)

Production of oil was approximately 45,000,000 bbl., or 5,000,000 bbl. less than in 1928. The fields near Tampico showed a decline of over 10,000,000 for both the light and heavy crudes, which was partly offset by an increase of over 5,000,000 bbl. in the Isthmus of Tehuantepec fields.

Petroleum Production in Dutch East Indies and Sarawak (Western Borneo) in 1929

By J. Th. Erb, The Hague, Netherlands

(Trans., Petrol. Devel. & Tech., 578. 450 words)

The total crude oil production of these islands was 5,838,253 metric tons (43,000,000 bbl.) in 1929, an increase of about 838,000 metric tons over 1928.

Petroleum Production in Rumania in 1929

(*Special Correspondence*)

(Trans., Petrol. Devel. & Tech., 579. 900 words)

PRODUCTION in 1929 was approximately 35,556,000 bbl., as against 31,542,000 bbl. in 1928. The increase in production was practically all accounted for in the District of Prahova, which produces over 70 per cent of all the Rumanian crude.

Review of Colombian Operations in 1929

By MICHAEL O'SHAUGHNESSY, New York, N. Y.

(Trans., Petrol. Devel. & Tech., 581. 1500 words)

THIS paper reports the petroleum legislative situation for 1929. The Tropical Oil Co. produced 20,384,548 bbl. of crude oil.

Petroleum Production in Argentina

By JOSE M. SOBRAL, Buenos Aires, Argentina

(Trans., Petrol. Devel. & Tech., 585. 200 words)

ARGENTINA produced 9,381,692 bbl. of oil in 1929.

Petroleum Development in Canada during 1929

By T. G. MADGWICK and WILLIAM CALDER, Ottawa, Ont.

(Trans., Petrol. Devel. & Tech., 586. 900 words)

PRODUCTION in 1929 totaled 1,130,159 bbl. as against 631,668 bbl. in 1928. Production for 1929 was distributed as follows: New Brunswick, 7,800 bbl.; Ontario, 125,000 bbl.; Alberta, 997,359 barrels.

Petroleum Developments in Bolivia in 1929

By GILBERT P. MOORE, New York, N. Y.

(Trans., Petrol. Devel. & Tech., 588. 900 words)

BOLIVIA still remains among the oil countries that have proved oil acreage but no production that is being marketed. No steps were taken to provide facilities for transport of Bolivian oil to market. The potential production from completed wells was practically unchanged from the 1928 figure of 6000 bbl. daily.

Developments in Refinery Technology during 1929

By A. D. DAVID, New York, N. Y.

(Trans., Petrol. Devel. & Tech., 590. 1500 words)

THE writer summarizes in simple language the developments in refinery technology in 1929.

Petroleum Engineering Education

By H. C. GEORGE, Norman, Okla.

(Trans., Petrol. Devel. & Tech., 594. 700 words)

THIS report of the round table discussion of Petroleum Engineering Education, which took place at the February, 1930 Meeting of the Petro-

leum Division, consists of the questions sent to about 60 of the leading engineers and educators in the petroleum industry, the oral discussion at the round table and a summary of the written discussion.

NONMETALLIC MINERALS

Recent Outstanding Developments in the Nonmetallic Mineral Industries

By Oliver Bowles, Washington, D. C.

(Min. & Met., January, 37. 3750 words)

Overproduction is the greatest problem of many of these industries. In its train follow price cutting, interrupted operation and maintenance costs for idle plants. Improvements have been made in equipment methods of treatment and in cooperative efforts.

Discussions as follows appear in February, 102: Recent Trends in the Uses of Refractories, by F. W. Davis, Newark, N. J.; Feldspar Used for Glass Manufacture in Canada and Phosphate Developments in British Columbia, Canada, by Hugh S. Spence, Ottawa, Ont.; Recent Developments of the Sodium Sulfate Deposits of Western Canada, by L. H. Cole, Ottawa, Ont. Additional discussion, July, 339.

Barite in California

By Walter W. Bradley, San Francisco, Cal.

(Tech. Pub. No. 266; Class H, Non-metallic Minerals, No. 13)

Barite resources of California are ample to supply the present market demands of the Pacific Coast, both for ground barite and for lithopone; and apparently they are ample for any considerable increase that may occur as the population and coast industries advance. Previous to 1920, when the first plant in California for preparation of lithopone began operations, it was not possible to dispose of anything but a high-grade, white, dry-grinding crude; but now with two plants for acid-treating available, there is a market for off-color grades of the crude mineral. With deep drilling in areas of high gas pressure, the demand for barite in oil-well mudding appears to be increasing in certain Californian oil fields.

Hydration Factors in Gypsum Deposits of the Maritime Provinces

By H. B. Bailey, Fredericton, N. B.

(Tech. Pub. No. 308; Class H, Non-metallic Minerals, No. 15)

This is a study of the processes by which anhydrite deposits in Nova Scotia have been altered to gypsum. The author introduces the terms "top hydration," "bottom hydration," "side hydration" and "interior hydration" to characterize the processes observed, and describes in some detail the mechanism by which they operate.

Status of Phosphate Industry of Western United States

By Frank Cole, Anaconda, Mont.
(Min. & Met., February, 104. 2000 words)

The territory covered in this discussion includes all the states west of the Mississippi River. The limited absorptive capacity of local markets limits the output but demand is growing. Concentrated products that can be shipped are being produced.

Limestone Quarrying at Northampton Plant of Atlas Portland Cement Co.

By L. James Boucher, Northampton, Pa.
(Tech. Pub. No. 272; Class H, Non-metallic Minerals, No. 14)

The Northampton, Pa., plant of the Atlas company is one of the largest and most progressively managed in the portland cement industry. The daily output is 20,000 bbl. of cement for the manufacture of which 5500 tons of raw material is mined by open-cut methods. Operations may be favorably compared with those conducted by metal-mining companies. Methods of stripping, drilling, blasting, loading, transportation, and crushing are described in detail. The cement rock and limestone are passed through rotary driers, pulverized in mills of the Huntington type, followed by tube mills. Revolving kilns fired by powdered coal produce the cement clinker which after cooling is ground in equipment similar to that used for the raw materials. A recent addition has been a "Compeb" mill, a sectionalized tube 9 ft. diameter and 50 ft. long. The author of this paper is assistant manager of the plant.

Quarrying of Limestone at Lime Spur, Montana

By P. F. Minister, Butte, Mont.
(Min. & Met., February, 108. 2700 words)

After a brief discussion of the history of the quarry and of its geology, this paper describes an ingenious system that might be called an underground glory hole and the problem of breaking the stone small enough but not too small.

Application of the Wire Saw in Marble Quarrying

By W. M. Weigel, St. Louis, Mo.
(Tech. Pub. No. 262; Class H, Non-metallic Minerals, No. 12)

The wire saw has been extensively used in slate quarrying but until recently few attempts have been made to apply it to other stones. This paper is an account of its experimental use in the marble quarry of the Saint Clair Marble Co., near Guion, Ark. The installation is extremely simple and requires a minimum amount of supervision. It is still in an experimental stage, as up to October, 1929, only four cuts had been completed, so that while it seems to be successful it is not

yet safe to say that it is entirely satisfactory. Conditions are ideal in this particular place, as the quarry face is open at both ends, so that there is no expense of preparing openings for the sheave standards. As the quarry face advances, the cuts will become much longer, and it is questionable whether a wire of the necessary length will have sufficient tension to complete the cut at the center to something near the same depth as at the ends, within a reasonable time, and also have sufficient reserve strength to allow it to be pulled through the cut. Costs are given.

Impact Mills for Grinding Fire Clay

(Min. & Met., May, 266. 1600 words)

DISCUSSION of paper on this subject by O. M. Tupper, Jr., that was presented at October, 1929, San Francisco Meeting and published in MINING AND METALLURGY (1929) 445.

Precious Stones

By GEORGE FREDERICK KUNZ, New York, N. Y.

(Min. & Met., January, 40. 1000 words)

SINCE 1927, the diamond market has been stable. Emeralds are being mined in the Transvaal and Namaqualand as well as in Colombia and Russia. Beryl crystals found in Maine are probably the greatest crystals of mineral known. A great sapphire, a ruby and a spinel have been found in Burma. Tourmalin, agates and imperial jade have been mined.

MINING GEOLOGY

Mining Geology in 1929

By R. J. COLONY, New York, N. Y.

(Min. & Met., January, 23. 3500 words)

REVIEW of changes in trend of geologic thought related to distribution, origin, structure, mineralogical and chemical features and processes of ore deposition; also advances in methods of field study and in general application of geology.

Minerals in a Power-controlled World

By H. FOSTER BAIN, New York, N. Y.

(Min. & Met., August, 392. 2500 words)

AN excerpt from an address before the World Power Conference at Berlin, dealing with the question of mineral reserves. The author believes that such reserves exist in considerable quantity and that their availability for use depends upon technology and economics.

Recent Progress in the Mineral Industry of South America

By Lester W. Strauss, New York, N. Y.

(Min. & Met., September, 428. 8400 words)

This paper shows the relative importance of the principal metallics and nonmetallics, including petroleum, produced in South America, and the countries that produce them. It gives statistics regarding the individual minerals.

A Geological Survey of California

By Walter W. Bradley and Olaf P. Jenkins, San Francisco, Cal.

(Min. & Met., November, 520. 1600 words)

In April, 1930, California State Division of Mines (formerly State Mining Bureau) observed its fiftieth anniversary. Its primary function is to make economic surveys of mineral resources, their development and utilization; also geological studies. Results are published in bulletins and annual reports. Fundamental geological survey has been begun; bibliography of Californian geology is in press. Twelve field parties were working during the past summer. Compilation of new geologic map is in progress. Excellent cooperation with U. S. Geological Survey, universities, railroad and other companies. Work being planned on basis of 10-year program.

Age and Structure of the Vein Systems at Butte, Montana

By James C. Ray, Stanford University, Cal.

(Tech. Pub. No. 265; Class I, Mining Geology, No. 27)

The author discusses the general theory of shearing and describes some experimental work. After describing the Butte system of veins in some detail he comes to the following conclusions, which are supported by the actual conditions in the Butte district. They reconcile and coordinate physiographic, structural and mineralogical data which, when considered independently, have sometimes appeared to offer contradictory evidence. 1. The three mineralized vein systems of the Butte district are of the same age and were formed during a single period of structural readjustment. 2. The observable faulting along the northwest and northeast vein systems could have taken place simultaneously. 3. The northwest and northeast vein systems were not continuous fractures at the time of their mineralization but were compression zones along which actual fractures existed only where minor intrafault-block adjustments caused comparatively slight movements. 4. The east-west (Anaconda) veins were formed in tension fractures which were the principal channels of circulation and from which were tapped such mineralizers as found their way into and formed the ores of the northwest and northeast veins. 5. Forces active during posthypogene mineralization formed the present continuity of the northwest and northeast veins. This continuity did not exist at the time of their hypogene mineralization or the existing barren brecciated zones along their present courses must have been

mineralized. 6. Hypogene mineralization occurred as a result of and in conjunction with regional diastrophism during the Oligocene period. 7. Regional diastrophism that occurred during the Pliocene and Pleistocene periods can well account for the posthypogene reopening and faulting of the mineralized vein systems as well as the formation of the later unmineralized faults of the Butte district. He concludes with these postulations: (a) A fracture or fissure is not a vein until such fracture or fissure has become mineralized. (b) The age of a vein should be determined by the age of its mineralization and not by the age of the fracture in which the mineralization takes place. (c) Intersecting fractures may be of different ages but if they all are mineralized at the same time the resulting fractures are of the same age.

Structure of Gold-bearing Quartz in Northern Ontario and Quebec

By George W. Bain, Amherst, Mass.

(Tech. Pub. No. 327; Class I, Mining Geology, No. 30)

Quartz is not essential to a gold deposit of this general type. Any rock containing at least some silica or silicate, and brittle enough to have open fissures, may carry gold. Many of the deeper parts of veins at Pearl Lake have no quartz and the gold occurs in fractures in disseminated sulfides. Some deposits have been formed above the critical temperature of the solutions that made them and no open channel for entrance of the solutions is visible. All gold, however, is introduced into exceedingly small fractures or planes of weakness in rock which has undergone some earlier stage of mineralization. Brittle quartz lying between walls strong enough to transmit crushing stresses to it is one of the more favorable places for development of planes of weakness and open fractures in which the late-stage minerals may be deposited. Quartz is a host mineral and forms no appreciable part of the products of crystallization from the solutions bringing in the gold. Pyrite forms an equally good host. Quartz is invariably dissolved along a gold-bearing fissure. Sericite or some salt of a partly neutralized base (mineral containing hydroxyl radical) invariably occurs in the gold-bearing fissure, or at some point along it. Pyrite or other sulfides are never dissolved along a gold-bearing fissure. The sulfides, therefore, can exert no reducing effect upon the gold-bearing solutions; they are hosts only. The reactions between the solutions and the quartz walls of the fractures, according to the author, seem to indicate that gold is deposited by neutralization of an alkaline basic solution, in which gold is known to be soluble, by silicon anhydride or a silicon acid derived from a silicate. Fractures in which gold and its associates occur are capillary or subcapillary in size, and the distribution of minerals shows that the solutions in most cases moved with normal hydraulic flow. Their surface energy must, therefore, have been high and their viscosity low; these conditions could exist only near or above their critical temperature.

Geology of the Parral Area of the Parral District, Chihuahua

By HARRISON SCHMITT, Hanover, N. M.

(Tech. Pub. No. 304; Class I, Mining Geology, No. 29)

JURASSIC limestones and shales were folded, uplifted and then eroded to form a mountainous topography upon which a thick series of volcanic rocks was deposited. Intrusions and faulting followed. Mineralizing solutions seem to have been deposited before the adjustment by faulting was complete. Sphalerite, galena, pyrite and argentite are the predominating "ore" minerals. The oreshoot along the Veta Colorado was possibly localized by breccia filled depressions in the foot wall of the Veta Colorado fault. The Palmilla ore deposits are at vein junctions, the structural relations and type of mineralization resembling those of the Comstock lode. Outcrop criteria for oreshoots in depth are the presence of silicification residual fluorite, residual oxidized lead minerals and some silver.

The Pao Deposits of Iron Ore in the State of Bolivar, Venezuela

By ERNEST F. BURCHARD, Washington, D. C.

(Tech. Pub. No. 295; Class I, Mining Geology, No. 28)

IN the Pao deposits in the State of Bolivar, Venezuela, it is estimated that at least 35,000,000 tons of iron ore is available, of which 15,000,000 is classified as "in sight." A large part of the deposit could be mined most economically by open cuts. The ore is predominantly hard, crystalline, hematite, but most of it carries magnetite irregularly distributed within the mass, or distributed in fairly definite bands. The analyses show an iron content ranging generally between 65 and 70 per cent. The phosphorus is generally below the Bessemer limit, and other deleterious elements such as titanium and sulfur likewise are low. The ore is suitable for making Bessemer steel and should also be suitable for smelting to pig iron or steel in the electric furnace. The proximity of large undeveloped water powers and extensive forests from which charcoal might be made suggest the possibility of local manufacture of iron and steel.

Occurrence of Quicksilver Orebodies

By C. N. SCHUETTE, San Francisco, Cal.

(Tech. Pub. 335; Class I, Mining Geology, No. 31)

THIS paper was presented at the New York meeting and read by title. About 50 of the 87 pages are devoted to the description of ore deposits in the United States and the remainder to those in foreign countries. The author concludes that given a region in which quicksilver occurs, the orebodies should be sought where conditions favoring a concentration of magmatic solutions exist. When viewed and judged from this standpoint the frantic efforts to class the orebodies as veins stockworks, beds, impregnations or whatnot are seen to be unnecessary. The

question of whether they will extend to great depth or not is generally irrelevant, as only in exceptional cases does this material affect the *extent* of the orebodies. A flat-lying orebody is preferable to one of similar extent dipping steeply and thus attaining depth, because the flat-lying orebody can be extracted through less expensive shafts than a vertical one. There is no inherent virtue in a quicksilver orebody merely because it goes to great depth. The age of the associated rocks and gangue minerals is more or less accidental. They are attendant but not causal occurrences. Examples of quicksilver orebodies in many different parts of the world have been cited. At some of these, as at Montebuono in Italy, opalite in Oregon and the B. & B. in Nevada, the mineralizing solutions reached the surface. Only a partial concentration of the primary mineralization took place and only low-grade orebodies were formed. Such surface-formed orebodies are generally marginal, depending on the price of the metal. Other examples, such as the Black Butte, Non Pareil and Bonanza in Oregon, the Castle Peak in Nevada and the Arizona mines, formed low-grade orebodies only because of the tightness of the receptacle rock. The largest and most productive mines, which had high-grade ore in quantity, are those in which the most favorable conditions for concentration existed. Fissures from the parent magma to the point of deposition exist. Impervious rock strata directed and limited the magmatic solutions to pervious rock masses. These receptacle rocks were breccias or sandstones of generous interstitial space. The ore mineral is the primary cinnabar precipitated in concentrated condition on account of loss of pressure and temperature of the primary alkaline thermal solutions. The shape of the orebody is determined by the disposition of the confining rocks.

The Unexpected in the Discovery of Orebodies

By Ira B. Joralemon, San Francisco, Cal.

(Tech. Pub. No. 340; Class I, Mining Geology, No. 32)

Many orebodies of great richness have been found unexpectedly. The northern Rhodesian copper deposits of B'wana M'Kubwa and Roan Antelope were practically worthless with 2 to 3 per cent oxidized copper ore down to 80 ft. in depth and were thought to be similar to the Union Minière du Haut Katanga oxidized ores. At this stage of affairs T. F. Field and H. H. d'Autremont visited Rhodesia for American interests in 1926. Purely by accidental microscopic observation these two engineers recognized minute disseminations of chalcocite ore in some of the sandstone at the 80 to 100 ft. depth. This was entirely unexpected; no surface indications were present to intimate secondary enrichment at depth.

Similarly the author pointed out the highlights referring to the unexpected discovery of the Colorado orebody at Cananea; the Frood orebody of International Nickel; the Campbell orebody at Bisbee; and the H orebody at Noranda, in Quebec.

Ore Deposition in Open Fissures Formed by Solution Pressure

By Alfred Wandke, Guanajuato, Gto., Mexico

(Tech. Pub. No. 342; Class I, Mining Geology, No. 33)

This paper directs attention to a process of ore emplacement which seems not to have received due consideration. Experimental work has indicated that ore solutions frequently depart from the parent magma under a pressure that would compel them to force apart the rock walls. Certain examples of flat-lying veins are cited which apparently show conclusively that this process has been operative. It is also concluded that certain vertical lenticular orebodies, where the proof is less conclusive, have nevertheless also been emplaced in openings formed by the pressure of the ore-bearing solutions.

MINING ADMINISTRATION

Governmental Control of the Production and Sale of Mineral Resources

By William E. Colby, San Francisco, Cal.

(Tech. Pub. No. 325; Class A, Metal Mining, No. 37; Class F, Coal and Coke, No. 34; Class G, Petroleum and Gas, No. 29; Class H, Non-metallic Minerals, No. 16; Class K, Mining Administration, No. 3)

Governmental control of mineral resources is complicated by the fact that both the Federal and State officials have certain powers. The Federal government has exclusive jurisdiction to determine the manner in which minerals shall be extracted and marketed from the public domain. The recent action in cancelling certain oil and gas prospecting permits and refusing to issue others until the present over-production has been checked is an instance of control.

The authority of the states arises out of a broad interpretation of "police power," the scope of which is expanding with the gradual changing of social and economic conditions, though the courts have been slow to uphold legislation aimed to control output and sale of minerals. Because of the migratory nature of oil and gas deposits, legislation involving control of them has been more successful. The ground has been that regulation has been necessary in the public interest to prevent over-production and waste.

The principle of the severance of minerals from surface ownership is the ideal system of law. President Hoover has proposed that surface rights of remaining public lands, subject to specified reservations, be transferred to State governments. The desirability of this policy from the standpoint of the miner is open to question.

Increased governmental control is inevitable, but there is a happy medium between drastic government paternalism and complete freedom of individual operation.

Progress Toward Security and Stability

By Herbert Hoover, Washington, D. C.
(Min. & Met., November, 508. 1100 words)

Excerpts from an address before the American Federation of Labor, Boston, Mass., Oct. 6, 1930.

Association Work

By J. William Wetter, Philipsburg, Pa.
(Min. & Met., November, 528. 800 words)

The author has observed that the work of many organizations is conducted by a "clique" which is always made up of aggressive men who will work for the success of the organization. The clique will welcome with open arms anyone who manifests an interest and signifies a desire to assist in the work.

Solving Distribution Problems by Merger

By Harold Vinton Coes, New York, N. Y.
(Min. & Met., November, 529. 5300 words)

A merger is primarily instigated for the resulting economies in distribution, administration, production and finance and the opportunity for meeting competition more successfully. The invasion of the coal market by competing commodities requires sound plans to meet the situation. It is quite conceivable that a merger in the coal industry would prove successful if organized with the proper units to produce coal of the various types required by a given market, to improve the preparation and merchandising methods, to ultimately reduce the cost of coal to the consumer, and to provide extensive research into further developments such as those for fuel-burning efficiency.

A Plea for a United States Court of Patent Appeals

By William Greenawalt and Kenneth W. Greenawalt, New York, N. Y.
(Min. & Met., February, 85. 7000 words)

A thoughtful discussion to establish the argument that technically trained men are necessary to hand down a sound judgment on most points involving science and technology.

Factors Affecting the Replacing of Equipment

By P. B. Bucky, New York, N. Y.
(Min. & Met., February, 99. 3500 words)

This paper gives mathematical formulas for determining (1) the desirability of replacing present equipment, and its corollary, the desirability of keeping present equipment, and (2) the date at which it will be desirable to replace present equipment. Discussion, May 267.

Employees' Clubs

(Min. & Met., July, 344. 1300 words)

DISCUSSION of clubs to enable employees to keep in touch with the work of the entire plant without interference with one another's department. The clubs at the Copper Queen mine and smelter are especially described.

GEOPHYSICAL PROSPECTING

Geophysical Prospecting in 1929

By DONALD H. McLAUGHLIN, Cambridge, Mass.
(Min. & Met., January, 26. 2900 words)

REVIEW of respective merits of methods and instruments. Need for research and instruction and for skillful interpretation is emphasized.

Geophysics Education

By C. A. HEILAND, Golden, Colo.
(Min. & Met., May, 250. 1200 words)

THE basis for training in geophysics should be geology, with extensive work in mathematics and physics.

Choice of Geophysical Methods

By FRANK RIEBER, San Francisco, Cal.
(Min. & Met., June, 301. 5400 words)

THE subject of this paper is the problem of checking the rising cost of discovery of new fields by the proper employment of geophysical methods, remembering that geophysical methods never yield final conclusions.

The Status and Importance of Isostasy

By WILLIAM BOWIE, Washington, D. C.
(Min. & Met., February, 93. 2600 words)

REVIEW of the tests of isostasy by the U. S. Coast and Geodetic Survey and in Canada, a review of information regarding gravity anomalies, and suggestions for research. Discussion in April, 226.

Interpretation of Gravitational Anomalies, II

By H. SHAW, London, England
(Tech. Pub. No. 338; Class L, Geophysical Prospecting, No. 21)

THIS is a continuation of the paper on this subject published as TECHNICAL PUBLICATION No. 178 last year. The first paper considered a heavy horizontal block extending to infinity, and a similar block with an inclined face. This paper considers the simple infinite vertical block and one inclined to the vertical, making the same general assumptions as in the first paper. The first part of the paper considers

the gradient characteristics, variation of maximum gradient with depth, position of the maximum gradient, variation of gradient with distance from midpoint, gradient value at definite distance from center line, inclination of the gradient curve at the origin, gradient interpretation, the characteristics of differential curvature, variation of maximum curvature with depth, position of maximum curvature, variation of curvature with distance from center line, position at which curvature attains definite values, the comparison of gradient and curvature effects, intersection of gradient and curvature curves, relation between gradient and curvature curves, relation between gradient and curvature values, combined gradient and curvature interpretation, geometrical construction and a comparison of horizontal and inclined blocks under the conditions assumed. The second part similarly takes up the infinite inclined block.

Depth of Investigation Attainable by Potential Methods of Electrical Exploration and Electrical Studies of the Earth's Crust at Great Depths

By Conrad and Marcel Schlumberger, Paris, France

(Tech. Pub. No. 315; Class L, Geophysical Prospecting, No. 18)

In May, 1928, electrical investigations at great depths were undertaken in the region of Vitré, Brittany. The purpose of these investigations was to ascertain the depths of investigation attainable by electrical prospecting and to study the resistivities of the rocks at very great depths. Measurements were made in cooperation with the Ministry of Posts and Telegraphs, and the length of lines utilized varied between 2 and 200 kilometers. This latter length corresponds to a 50-km. depth.

The paper indicates how several difficulties encountered in the course of the investigation were overcome—in particular, the difficulties resulting from the skin effect and from the earth currents—and the results obtained regarding the skin effect and the resistivity of the rocks at great depths are given.

Electrical Prospecting for Ore and Oil

By Hans Lundberg, New York, N. Y.

(Min. & Met., April, 210. 2400 words)

A concise summary of the principal factors to be considered in deciding upon the method most likely to give results.

Observed and Theoretical Electromagnetic Model Response of Conducting Spheres

By L. B. Slichter, Madison, Wis.

(Tech. Pub. No. 332; Class L, Geophysical Prospecting, No. 20)

After statement of the principles of similitude which apply to electromagnetic modeling, charts showing the inductive response of conducting spheres as dependent on frequency, conductivity, and size are exhibited

in this paper. The shape of the curve, which relates intensity of response to frequency, is significant in indicating the advisability of low frequencies in the exploration for conducting bodies, when consistent with the securing of a large percentage of the ideally possible response. Comparisons of theoretical respnse and that observed with models are shown, and relationships with the subject of applied electromagnetic prospecting are indicated.

Absorption of Electromagnetic Induction and Radiation by Rocks

By A. S. Eve, Montreal, Que.

(Tech. Pub. No. 316; Class L, Geophysical Prospecting, No. 19)

In prospecting for orebodies underground by electrical methods the usual practice is to employ a loop of wire carrying alternating current that induces an electromotive force in good conductors beneath the surface. One of the factors in the problem is the absorption of the electrômagnetic disturbances by the intervening soils and rocks. The theory on radiation waves is summarized and the results of a series of experiments made in the Mammoth Cave in Kentucky are outlined. The cave was because of the absence of metallic conductors, the "overburden" being horizontal layers of sandstone and limestone. The limestone had a resistivity of 100,000 ohm-cm. It appears that Morse signals can be picked up readily in underground mine workings with receiving coils and head phones as the only apparatus.

Mapping Oil Structures by the Sundberg Method

By Theodor Zuschlag, New York

(Tech. Pub. No. 313; Class L, Geophysical Prospecting, No. 17; Class G, Petroleum and Gas, No. 26)

The basic principle of electromagnetic prospecting in its application to geologic structure is the measurement of the electromagnetic field resulting from currents set up in strata which, due to their content of saline solutions, are of high electrical conductivity. This permits the determination of the depth to these strata, and consequently their elevation can be determined at a sufficient number of points to give their dip and strike and thus determine the form of geologic structure present. Following a brief non-technical discussion of the principles involved, diagrams of the electrical indications resulting from typical geological structures are given, followed by maps showing the actual results of electrical surveys over a salt dome and a faulted area respectively. These results are then compared with the evidence derived from drilling, and the electrical prospecting results are shown to accord with the geology.

Practical Geomagnetic Exploration with the Hotchkiss Superdip

By Noel H. Stearn, St. Louis, Mo.

(Tech. Pub. No 370; Class L, Geophysical Prospecting, No. 24)

To assist mining engineers and economic geologists who find it necessary to categorize the various geophysical methods and instruments

this paper discusses the place of the Hotchkiss Superdip. The principle, construction, behavior, sensitivity, manipulation, field procedure and corrections are outlined in such a way as to be understood by the non-specialist.

The author concludes: "In categorizing the Superdip, however, it should be borne in mind that the instrument has been designed as a facile tool for the use of mining engineers and economic geologists, and that its use is recommended for magnetic anomalies of an order of magnitude greater than 10 gammas."

Seismic Propagation Paths

By Maurice Ewing and L. Don Leet

(Tech. Pub. No. 267; Class L, Geophysical Prospecting, No. 16)

Assuming that wave velocities in seismic prospecting increase as a continuous linear function of the depth, the authors have derived formulas for computing, from two time-distance observations, the amount of velocity increase, depth of penetration, and a graphical determination of the path of the vibrations and have discussed the ground, reflected and refracted waves. The literature of the subject is first reviewed, and then the authors proceed to the derivation of their own formulas. Then numerical examples are given to illustrate the application of the formulas. The paper concludes with a two-page bibliography.

A New Geophone

By C. A. Heiland, Golden, Colo.

(Tech. Pub. No. 330; Class L, Geophysical Prospecting, No. 22)

The improved geophone described in this paper was devised by the author and C. H. Hull. After describing the work done on the geophone during the war and that by the Bureau of Mines subsequent to that time, the author discusses the possibility of the use of it for geophysical prospecting, and discussing the range of its sensitivity the author describes his method of magnification of the amount of air movement by the aid of a reflected beam of light.

Summaries of Results from Geophysical Surveys at Various Properties

(Tech. Pub. No. 369; Class L. Geophysical Prospecting, No. 23)

This pamphlet contains a series of summaries of geophysical work done on a number of properties in the United States and Canada, with one covering briefly the Bushveld, South Africa. There is an introduction by D. H. McLaughlin, chairman of the Committee on Geophysical Prospecting and a general discussion of the summaries.

MISCELLANEOUS

Engineering Education

By THEODORE J. HOOVER, Stanford University, Cal.

(Min. & Met., February, 74. 1100 words)

EXCERPTS from an address delivered at joint meeting of the San Francisco Section and the Stanford Section, Mining and Geological Society of American Universities.

Mineral Education in 1929

By E. A. HOLBROOK, Pittsburgh, Pa.

(Min. & Met., January, 32. 1000 words)

BOTH the name and scope of work in schools of mines are being broadened and there is a paradoxical trend toward and away from greater specialization.

The Changing Field in Metallurgical Education

By DAVID F. MCFARLAND, State College, Pa.

(Min. & Met., May, 246. 1800 words)

DESCRIBES the growth in curricula from the days of "metallurgical options" to the present broad courses which include much special fundamental knowledge other than engineering.

What Does Industry Want in the Training of Metallurgists?

By STEPHEN L. GOODALE, Pittsburgh, Pa.

(Min. & Met., May, 248. 1900 words)

DIGEST of replies to a questionnaire, with the concluding statement by the author that the University of Pittsburgh is trying to comply with the demand of industry that students shall be given courses to provide the most thorough training possible in the fundamental sciences, some cultural study, some business law and finance and all the metallurgy that can be crowded into the four-year course.

Why Do Few Students Elect Metallurgy?

By CHARLES Y. CLAYTON, Rolla, Mo.

(Min. & Met., May, 251. 1350 words)

THE author answers this question by showing that metallurgy is a word known to few outside of technical fields, partly because metallurgy has been looked upon as a simple branch of applied chemistry. He urges more publicity for the opportunities in metallurgical work.

Metallurgical Laboratories

By Carle R. Hayward, Cambridge, Mass.
(Min. & Met., May, 252. 3000 words)

The general aspects of college laboratory work are illustrated by a description of the laboratories at the Massachusetts Institute of Technology.

Research

By Charles M. A. Stine, Wilmington, Del.
(Min. & Met., May, 261. 2400 words)

The director of development in an organization that has been a leader in building business on research and its application discusses its opportunities and its rewards.

Making the Mining Industry More Attractive to the Graduate

By Hillary W. St. Clair, Salt Lake City, Utah
(Min. & Met., August, 385. 2200 words)

A statement of the problem as seen by an undergraduate at the University of Utah.

Mining and Manufacturing

By M. S. North, Chicago, Ill.
(Min. & Met., November, 534. 750 words)

Mining and manufacturing have long been considered separate and distinct enterprises. Differences that exist are, in the case of the mines, the result of adhering to past practices which were sound but do not measure up to modern business practice. It is true that mining has developed processes purely technical that indicate great progress but usually in the engineering field. This progress may be likened to the progress in the development of specialized machines in factories. Methods of management, merchandising, cost finding, budgeting and the elimination of waste of men, materials and machines have not received the consideration usually given in modern up-to-date factories. The traditional difference between mining and manufacturing is of lesser importance today than ever before. Mining is modernizing but great advancement must yet be made to bring into play all fundamental economics appreciated in and made a part of modern manufacture.

A Bird's-eye View of South America

By Corey C. Brayton, Oakland, Cal.
(Min. & Met., November, 525. 3000 words)

The writer describes a six months' business trip made last winter and spring for the American Manganese Steel Co., which gave him and his son a very satisfactory bird's-eye view of the Canal Zone and South-America. They saw all the great mining properties and their itinerary

included the platinum dredging in "Jungle-Land," and the Maracaibo oil field, as well as the thriving East Coast and the West Indies. What they saw clearly indicated the usual versatility and adaptability of the mining engineer.

South America is a tremendous country of rapid development along modern lines. Airplane transportation is the latest and most outstanding accomplishment, for distances are long. The consolidated system now covers some 15,000 miles of both coasts and cross country. In addition there are several local lines. The writer and his son made some 8000 miles of the journey, including the crossing of the Andes, by air.

Sanitary Protection at Mining Camps

By E. B. Besselievre, New York, N. Y.
(Min. & Met., November, 538. 2700 words)

Stresses the importance of sanitation of mining projects in general and describes sewage-treatment plants at the town sites of Noranda, Quebec, and Flin Flon, Manitoba, which are modern plants equipped with Dorr units similar to those in municipal use elsewhere. Conditions are severe at Noranda, as the water supply and point of final discharge of the sewage-plant effluent are the same body of water. The paper also describes the water-purification plant at Noranda. Health authorities of the Province of Quebec give Noranda water the highest rating in the province. There have been no epidemics of typhoid at either camp, as at Cobalt.

Early Day San Francisco Stock Market Fluctuations on the Mines of the Comstock Lode, Nevada

By Henry Lawrence Slosson, San Francisco, Cal.
(Min. & Met., February, 110. 1100 words)

Had it not been for this wild era of speculation it is doubtful whether the lode would have ever been developed, particularly after the rich surface orebodies had been mined out and the persistence of ore in depth had been doubted by some very eminent geologists.

Metal Prices

By Frederick W. Bradley, San Francisco, Cal.
(Min. & Met., December, 572. 3800 words)

A discussion of fluctuation in commodity prices after some of the great wars of history, followed by special discussion of nonferrous metals and the possibility of reasonable stabilization.

Is the Producer of Gold a Social Parasite?

By Zay Jeffries, Cleveland, Ohio
(Min. & Met., December, 571. 1000 words)

About half the gold produced since the discovery of America has been largely consumed for useful things or for purposes which otherwise gave

satisfaction to human beings. More than half the gold producers since America was discovered can therefore be removed from the "parasite" charge. The balance of the gold is accounted for as the foundation for world credit and commerce. There are required for the production of credit gold approximately 250,000 people—one person in about 7000, or about 0.015 per cent of the world population. The author asks, Who can suggest a way to accomplish the involved and difficult function of credit gold with as little expenditure of human energy? The producer of credit gold, far from being a social parasite, renders mankind one of its most useful and valuable services.

Can Silver Come Back?

By W. F. Boericke, New York, N. Y.

(Min. & Met., April, 207. 3200 words)

Brief analysis of present demand and inquiry into stimulation of industrial uses.

Suggested Solution of the Silver Problem

By Harrington Emerson, New York, N. Y.

(Min. & Met., December, 589. 2300 words)

Advocates adoption by the United States of a silver policy of its own, making silver an alternate money metal. Domination of the ratio between gold and silver prices per ounce has been held by various countries, and the United States should hold that domination now. It should abolish the silver dollar and should make silver certificates redeemable in silver bullion at the posted price for the day. This is not the bi-metal standard. There is an absolute and clearly defined difference between a double standard and an alternate. The author wants the alternate right to use silver bullion by special contract; to pay taxes in gold or in silver bullion. Such legislation would stimulate activities in every mining camp—gold, silver, lead or copper; would increase the price of wheat and its alternates, and would restore the purchasing power of Japan, China and India.

Unwise and Dangerous Provisions of Engineering Registration Laws

By G. M. Butler, Tucson, Ariz.

(Min. & Met., March, 179. 3800 words)

Discussion of licensing laws by one who has had experience in this administration. The author urges mining engineers to be alert to prevent the passage of laws that will inconvenience them and restrict their freedom to choose their working places. Discussion in April, 225 and May, 277.

A. I. M. E. Technical Publications, 1930

[Separates of all the *Technical Publications* published in 1930 are available at Institute headquarters. All the papers are on file in public, university and technical libraries, and when so indicated in the list, may be found in the TRANSACTIONS.]

METAL MINING

TECHNICAL PUBLICATION No.	TITLE	AUTHOR	MEETING	TRANS. VOLUME AND PAGE*
264-A.29	Quicksilver Industry in 1929. Improvements in the Metallurgy of Quicksilver and Symposium on The Present Status of the Quicksilver Industry (With Discussion) (37)	L. H. Duschak C. G. Maier and others	San Francisco Oct., 1929	**T30**, 283, *312*
271-A.30	Observation on Ground Movement and Subsidences at Rio Tinto Mines, Spain (20)	R. E. Palmer	New York Feb., 1930	**T30**, 168
274-A.31	Development and Installation of the Hawkesworth Detachable Bit (29)	C. L. Berrien	New York Feb., 1930	**T30**, 139, *166*
276-A.32	Protective Measures Against Gas Hazards at United Verde Mine (11)	O. A. Glaeser	New York Feb., 1930	**T30**, 129, *137*
287-A.33	Use and Cost of Compressed Air (12)	R. S. Lewis	New York Feb., 1930	
314-A.34	Miami Copper Company Method of Mining Low-grade Orebody (44)	F. W. Maclennan	New York Feb., 1930	**T30**, 39, *80*
319-A.35	How Human Beings Respond to Changing Atmospheric Conditions (9)	W. J. McConnell	New York Feb., 1930	
324-A.36	Vertical and Incline Shaft Sinking at North Star Mine (21)	Arthur B. Foote	San Francisco Oct., 1929	**T30**, 87
325-A.37	Governmental Control of the Production and Sale of Mineral Resources (With Discussion) (16)	W. E. Colby	New York Feb., 1930	
333-A.38	Some Recent Developments in Open-pit Mining on the Mesabi Range (25)	E. E. Hunner	New York Feb., 1930	**T30**, 106, *127*
334-A.39	Mining Methods and Costs at Presidio Mine of American Metal Co. of Texas (15)	V. D. Howbert R. Bosustow	El Paso Oct., 1930	
339-A.40	Operation of Pressure Fans in Series (8)	W. S. Weeks V. S. Grishkevich	New York Feb., 1931	
364-A.41	Top Slicing with Filling of Slices as Used at the Charcas Unit of Compania Minera Asarco, S. A. (17)	H. Willey	El Paso Oct., 1930	
373-A.42	Drill Sampling and Interpretation of Sampling Results in the Copper Fields of Northern Rhodesia (17)	H. T. Matson G. A. Wallis	New York Feb., 1931	

* Volume numbers are in boldface; page numbers of papers, Roman; and page numbers of discussions, italics. Where no number is given, paper has not appeared in TRANSACTIONS. Discussions are printed following the respective papers.

The significance of the volume numbers is given on page 427.

The number in parentheses following each title indicates the number of pages in the *Technical Publication*.

Milling and Concentration

Technical Publication No.	Title	Author	Meeting	Trans. Volume and Page
275-B.27	Classifier Efficiency; an Experimental Study (14)	A. W. Fahrenwald	New York Feb., 1930	B30, 82
312-B.28	Chemical Reactions in Flotation (33)	A. F. Taggart T. C. Taylor A. F. Knoll	New York Feb., 1930	B30, 217, *247*
326-B.29	A Laboratory Investigation of Ball Milling (26)	A. M. Gow A. B. Campbell W. H. Coghill	San Francisco Oct., 1929	B30, 51, *74*
368-B.30	Milling Methods and Costs at Presidio Mine of The American Metal Co. of Texas (20)	V. D. Howbert F. E. Gray	El Paso Oct., 1930	
371-B.31	Milling Practice at San Francisco Mines of Mexico, Ltd. (24)	G. L. Allen	El Paso Oct., 1930	

Iron and Steel

Technical Publication No.	Title	Author	Meeting	Trans. Volume and Page
268-C.41	Electrolytic Iron from Sulfide Ores. (35)	R. D. Pike G. H. West L. V. Steck R. Cummings B. P. Little	New York Feb., 1930	C30, 311, *343*
269-C.42	Tensile Properties of Rail and Other Steels at Elevated Temperatures (48)	J. R. Freeman, Jr. G. W. Quick	New York Feb., 1930	C30, 225, *270*
273-C.43	Production and Some Properties of Large Iron Crystals (17)	N. A. Ziegler	New York Feb., 1930	C30, 209, *224*
274-C.44	Development and Installation of the Hawkesworth Detachable Bit (29)	C. L. Berrien	New York Feb., 1930	T30, 139, *166*
280-C.45	Rate of Carbon Elimination and Degree of Oxidation of Metal Bath in Basic Open-hearth Practice. II. (7)	A. L. Feild	New York Feb., 1930	C30, 23, *38*
284-C.46	Sintering Limonitic Iron Ores at Ironton, Minn. (12)	P. G. Harrison	New York Feb., 1930	C30, 346, *355*
294-C.47	Endurance Properties of Steel in Steam (13)	T. S. Fuller	New York Feb., 1930	C30, 280, *292*
296-C.48	Production of Gray Iron from Steel Scrap in the Electric Furnace (17)	T. F. Baily	New York Feb., 1930	C30, 64, *78*
299-C.49	Influence of Rate of Cooling on Dendritic Structure and Microstructure of Some Hypoeutectoid Steel (16)	A. Sauveur C. H. Chou	New York Feb., 1930	C30, 100, *113*
309-C.50	Progress Notes on Iron-silicon Equilibrium Diagram (33)	B. Stoughton E. S. Greiner	New York Feb., 1930	C30, 155, *185*
310-C.51	Experiments Demonstrate Method of Producing Artificial Manganese Ore (29)	T. L. Joseph E. P. Barrett C. E. Wood	New York Feb., 1930	C30, 378, *404*
311-C.52	A New Method for Determining Iron Oxide in Liquid Steel (13)	C. H. Herty, Jr. J. M. Gaines, Jr. H. Freeman M. W. Lightner	New York Feb., 1930	C30, 28, *38*
331-C.54	The Future of the American Iron and Steel Industry (16)	Z. Jeffries	New York Feb., 1930	C30, 9
337-C.55	Permanent Growth of Gray Cast Iron (11)	W. E. Remmers	New York Feb., 1931	
347-C.53	Practical Observations on Manufacture of Basic Open-hearth, High-carbon Killed Steel (16)	W. J. Reagan	Chicago Sept., 1930	C30, 45, *58*
348-C.56	Transformation of Austenite at Constant Subcritical Temperatures (30)	E. S. Davenport E. C. Bain	Chicago Sept., 1930	C30, 117, *144*
355-C.57	Development of Casing for Deep Wells; a Study of Structural Alloy Steels (16)	F. W. Bremmer	Chicago Sept., 1930	C30, 293, *306*
372-C.58	Resistance of Iron Ores to Decrepitation and Mechanical Work (15)	T. L. Joseph E. P. Barrett	Chicago Sept., 1930	C30, 365, *413*

Non-Ferrous Metallurgy

Technical Publication No.	Title	Author	Meeting	Trans. Volume and Page
264-D.22	Quicksilver Industry in 1929. Improvements in the Metallurgy of Quicksilver and Symposium on the Present Status of the Quicksilver Industry (With Discussion) (37)	L. H. Duschak C. G. Maier and others	San Francisco Oct., 1929	T30, 283, *312*
279-D.23	Progress in the Production and Use of Tantalum (6)	G. W. Sears	New York Feb., 1930	T30, 317, *320*
303-D.24	Lead Refining at Bunker Hill Smelter of Bunker Hill & Sullivan Mining & Concentrating Co. (9)	A. F. Beasley	New York Feb., 1930	T30, 265, *271*
305-D.25	A Petrographic Study of Lead and Copper Furnace Slags (20)	R. D. McLellan	New York Feb., 1930	T30, 244, *261*
320-D.26	Electrolytic Cadmium Plant of Anaconda Copper Mining Company at Great Falls, Mont. (7)	W. E. Mitchell	New York Feb., 1930	T30, 239, *243*
321-D.27	Investigation of Anodes for Production of Electrolytic Zinc (8)	H. R. Hanley C. Y. Clayton D. F. Walsh	New York Feb., 1930	T30, 275, *281*
350-D.28	Leaching Process at Chuquicamata, Chile (55)	C. W. Eichrodt	New York Feb., 1930	T30, 186, *238*

Institute of Metals Division

Technical Publication No.	Title	Author	Meeting	Trans. Volume and Page
263-E.91	Stress-corrosion Cracking of Annealed Brasses (15)	A. Morris	New York Feb., 1930	E30, 256, *268*
270-E.93	A Theory Concerning Gases in Refined Copper (15)	A. E. Wells R. C. Dalzell	New York Feb., 1930	E30, 349, *361*
278-E.105	Working Properties of Tantalum (8)	M. M. Austin	New York	E30, 551
281-E.94	Effects of Oxidation and Certain Impurities in Bronze (17)	J. W. Bolton S. A. Weigand	New York Feb., 1930	E30, 368, *382*
282-E.95	Melting and Casting Some Gold Alloys (20)	E. A. Capillon	New York Feb., 1930	E30, 439, *456*
283-E.96	Oxides in Brass (19)	O. W. Ellis	New York	E30, 316, *332*
285-E.97	Certain Types of Defects in Copper Wire Caused by Improper Dies and Drawing Practice (14)	H. C. Jennison	New York Feb., 1930	E30, 121, *132*
286-E.98	Recent Developments in Melting and Annealing Non-ferrous Metals (16)	R. M. Keeney	New York Feb., 1930	E30, 414, *427*
288-E.99	The Alpha-beta Transformation in Brass (9)	A. J. Phillips	New York Feb., 1930	E30, 194, *200*
289-E.100	Comparison of Copper Wire Bars Cast Vertically and Horizontally (11)	J. W. Scott L. H. DeWald	New York Feb., 1930	E30, 338, *347*
290-E.101	Distribution of Lead Impurity in a Copper-refining Furnace Bath (5)	J. W. Scott L. H. DeWald	New York Feb., 1930	E30, 334, *336*
291-E.102	Thermal Conductivity of Copper Alloys, I.—Copper-zinc Alloys (24)	C. S. Smith	New York Feb., 1930	E30, 84, *105*
292-E.103	Alpha-phase Boundary of the Ternary System Copper-silicon-manganese (32)	C. S. Smith	New York Feb., 1930	E30, 164, *193*
293-E.104	Corrosion of Alloys Subject to the Action of Locomotive Smoke (16)	F. L. Wolf	New York Feb., 1930	E30, 219
297-E.106	Internal Stress and Season Cracking in Brass Tubes (18)	D. K. Crampton	New York Feb., 1930	E30, 233, *248*
298-E.107	Monel Metal and Nickel Foundry Practice (9)	E. S. Wheeler	New York Feb., 1930	E30, 458, *464*
300-E.108	Influence of Silicon in Foundry Red Brasses (18)	H. M. St. John G. K. Eggleston T. Rynalski	New York Feb., 1930	E30, 384, *399*
301-E.109	Directed Stress in Copper Crystals (26)	C. H. Mathewson K. R. Van Horn	New York Feb., 1930	E30, 59, *82*

Institute of Metals Division.—*Continued*

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302-E.110	Melting Bearing Bronze in Open-flame Furnaces (7)	E. R. Darby	New York Feb., 1930	E30, 407, *412*
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